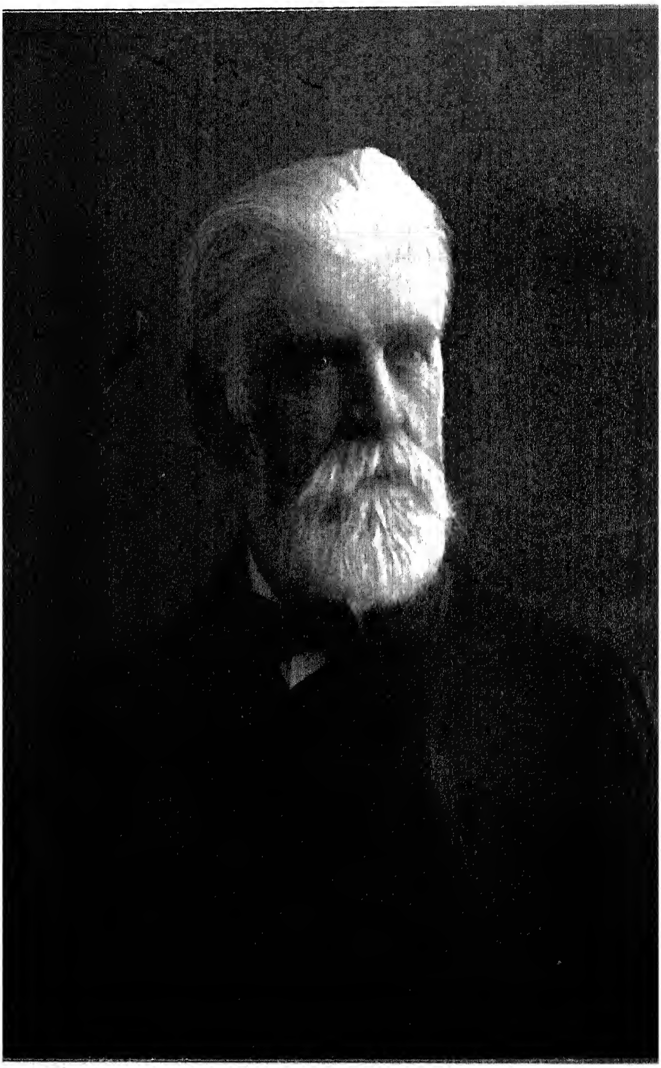

Wm P Blake

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XLI.

CONTAINING THE PAPERS AND DISCUSSIONS OF 1910.

NEW YORK, N. Y.:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1911.

PREFACE.

THIS volume contains all the proceedings, papers, and discussions of the Institute published during 1910, with the following exceptions:

1. Brief obituary notices of members and associates reported as deceased during the year 1910; library accessions and requirements; notices of meetings of the Institute and of other societies; lists of proposed members and associates; changes of address of members; and other announcements of general but temporary interest, furnished to members in *Bulletin* Nos. 37 to 48, during the year 1910.

2. Account of the excursions and entertainments connected with the Pittsburg meeting, March, 1910,[1] and with the Canal Zone meeting, November, 1910.[2]

3. Year Book, containing a List of Members and Associates, revised to Jan. 1, 1911, 173 pages. Published in separate form and distributed with *Bulletin* No. 49, January, 1911.

4. The following papers, presented at the Canal Zone meeting:

The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits in the United States, by William H. Emmons, Chicago, Ill.[3]

The Report of the Committee on Uniform Mining-Laws for the Prevention of Mine-Accidents, by W. R. Ingalls, J. Parke Channing, James Douglas, and John Hays Hammond, New York, N. Y., and James R. Finlay, Goldfield, Nev.[4]

Report of the Delegates of the Institute Attending the Convention of the American Mining Congress at Los Angeles, by D. W. Brunton, Denver, Colo.[5]

The Panama Canal. A general discussion.[6]

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[1] *Bulletin* No. 40, April, 1910, pp. 311 to 334.
[2] *Idem*, No. 48, December, 1910, pp. 1017 to 1054.
[3] *Idem*, No. 46, October, 1910, pp. 767 to 837.
[4] *Idem*, pp. 839 to 888.
[5] *Idem*, No. 48, December, 1910, pp. v to xi.
[6] *Idem*, No. 49, January, 1911, pp. 35 to 118.

5. A few discussions referring to papers contained in Vol. XL., which were received early in the year 1910, yet in time to be included in said volume.

On the other hand, this volume includes:

6. The following paper presented at the Spokane meeting, which was omitted from Vol. XL. on account of lack of space:

The Ventilating-System at the Comstock Mines, Nevada, by George J. Young, Reno, Nev.

JOSEPH STRUTHERS,
Secretary and Editor.

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PROCEEDINGS.

PAPERS.

DISCUSSIONS.

OFFICERS.

For the year ending February, 1911.

COUNCIL.*

PRESIDENT OF THE COUNCIL.

D. W. BRUNTON .... DENVER, COLO.

(Term expires February, 1911.)

VICE-PRESIDENTS OF THE COUNCIL.

W. C. RALSTON .... SAN FRANCISCO, CAL.
W. L. SAUNDERS .... NEW YORK, N. Y.
H. V. WINCHELL .... ST. PAUL, MINN.

(Term expires February, 1911.)

BENJAMIN B. LAWRENCE .... NEW YORK, N. Y.
JOSEPH W. RICHARDS .... SOUTH BETHLEHEM, PA.
ALBERT SAUVEUR .... CAMBRIDGE, MASS.

(Term expires February, 1912.)

COUNCILORS.

ARTHUR S. DWIGHT .... NEW YORK, N. Y.
R. V. NORRIS .... WILKES-BARRE, PA.
WILLIAM H. SHOCKLEY .... TONOPAH, NEV.

(Term expires February, 1911.)

KARL EILERS .... NEW YORK, N. Y.
ALEX. C. HUMPHREYS .... NEW YORK, N. Y.
W. G. MILLER .... TORONTO, CANADA.

(Term expires February, 1912.)

ROBERT E. JENNINGS .... NEW YORK, N. Y.
WILLIAM KELLY .... VULCAN, MICH.
CHARLES F. RAND .... NEW YORK, N. Y.

(Term expires February, 1913.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND, 29 W. 39th St. .... NEW YORK, N. Y.

(Term expires February, 1911.)

ASSISTANT SECRETARY AND EDITOR.

JOSEPH STRUTHERS .... NEW YORK, N. Y.

CORPORATION.

JAMES GAYLEY, President; JAMES DOUGLAS, Vice-President;
R. W. RAYMOND, Secretary; FRANK LYMAN, Treasurer;
JOSEPH STRUTHERS, Assistant Secretary and Assistant Treasurer.

DIRECTORS.

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.

(Term expires February, 1911.)

THEODORE DWIGHT, CHARLES H. SNOW, R. W. RAYMOND.

(Term expires February, 1912.)

JAMES GAYLEY, CHARLES KIRCHHOFF, FRANK LYMAN.

(Term expires February, 1913.)

Consulting Attorneys, Blair & Rudd, New York, N. Y.

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* SECRETARY'S NOTE.—The Council is the professional body, having charge of the election of members, the holding of meetings (except business meetings), and the publication of papers, proceedings, etc. The Board of Directors is the body legally responsible for the business management of the Corporation, and is therefore, for convenience, composed of members residing in New York.

OFFICERS ELECTED AT ANNUAL MEETING, FEB. 21, 1911.

The list of officers on the opposite page is for the year 1910, the period covered by the contents of this volume of the *Transactions*. But the result of the election at the Annual Business Meeting, February, 1911, although strictly belonging to the next volume, is here published for the convenience of members.

The following officers were elected by vote of the members and associates in person or by proxy at the Annual Meeting, Feb. 21, 1911:

COUNCIL.

PRESIDENT OF THE COUNCIL.

CHARLES KIRCHHOFF, ... New York, N. Y.
(To serve for one year. Term expires February, 1912.)

VICE-PRESIDENTS OF THE COUNCIL.

S. B. CHRISTY, ... Berkeley, Cal.
W. A. LATHROP, ... Philadelphia, Pa.
GARDNER F. WILLIAMS, ... Washington, D. C.
(To serve for two years Term expires February, 1913.)

COUNCILORS.

A. E. CARLTON, ... Cripple Creek, Colo.
W. J. OLCOTT, ... Duluth, Minn.
E. L. YOUNG, ... New York, N. Y.
(To serve for three years. Term expires February, 1914.)

SECRETARY OF THE COUNCIL.

R. W. RAYMOND* (JOSEPH STRUTHERS), . . New York, N. Y.
(To serve for one year. Term expires February, 1912.)

ASSISTANT SECRETARY AND EDITOR (BY APPOINTMENT).

JOSEPH STRUTHERS, ... New York, N. Y.
(To serve for one year. Term expires February, 1912.)

DIRECTORS OF THE CORPORATION.

JAMES DOUGLAS, JAMES F. KEMP, ALBERT R. LEDOUX.
To serve for three years. Term expires February, 1914.)

The following are the officers of the Corporation for the year ending February, 1912:

President, James Gayley, New York, N. Y.
Vice-President, James Douglas, New York, N. Y.
Secretary, R. W. Raymond, New York, N. Y.* (Joseph Struthers).
Treasurer, Frank Lyman, New York, N. Y.
Assistant Treasurer, Joseph Struthers, New York, N. Y.

* Resigned Mar. 31, 1911, and Joseph Struthers appointed to fill vacancy.

PAST OFFICERS.

Presidents.

*David Thomas	1871
R. W. Raymond	1872–1874
*A. L. Holley	1875
*Abram S. Hewitt	1876
*T. Sterry Hunt	1877
*Eckley B. Coxe	1878–1879
*William P. Shinn	1880
*William Metcalf	1881
*Richard P. Rothwell	1882
Robert W. Hunt	1883
James C. Bayles	1884–1885
Robert H. Richards	1886
*Thomas Egleston	1887
William B. Potter	1888
Richard Pearce	1889
*Abram S. Hewitt	1890
John Birkinbine	1891–1892
H. M. Howe	1893
John Fritz	1894
*J. D. Weeks	1895
E. G. Spilsbury	1896
*Thomas M. Drown	1897
Charles Kirchhoff	1898
James Douglas	1899–1900
E. E. Olcott	1901–1902
Albert R. Ledoux	1903–1904
James Gayley	1905
Robert W. Hunt	1906
John Hays Hammond	1907–1908
D. W. Brunton	1909–1910
James Gayley (Corporation)	1905–1910

Secretaries.

*Martin Coryell	1871–1872
*Thomas M. Drown	1873–1884
R. W. Raymond	1884 ——

Treasurers.

J. Pryor Williamson	1871–1872
*Theodore D. Rand	1872–1903
Frank Lyman	1903 ——

* Deceased.

HONORARY MEMBERS.

Year of Election		
1876.	Prof. Richard Åkerman	Stockholm, Sweden.
1909.	Prof. Richard Beck	Freiberg, Germany.
1905.	Andrew Carnegie	New York, N. Y.
1906.	Dr. James Douglas	New York, N. Y.
1888.	Prof. Haton de la Goupillière	Paris, France
1906.	Sir Robert A. Hadfield	London, England
1888.	Prof. Hans Hoefer	Leoben, Austria.
1905.	Prof. Henri Louis Le Chatelier	Paris, France.
1899.	M. Floris Osmond	Saint Leu, France.
1909.	Alexandre Pourcel	Paris, France.
1909.	Dr. Ing. H. C. Emil Schroedter	Düsseldorf, Germany.
1906.	John E. Stead	Middlesbrough, England.
1909.	James M. Swank (Associate)	Philadelphia, Pa.
1902.	Prof. Dimitry Constantin Tschernoff	St. Petersburg, Russia.
1910.	Prof. Tsunashiro Wada	Tokyo, Japan.
1907.	Charles D. Walcott	Washington, D. C.

Honorary Members (*Deceased*).

		Year of Decease.
1872.	Bell, Sir Lowthian	1904
1892.	Castillo, A. Del	1895
1902.	Contreras, Manuel Maria	1902
1888.	Daubrée, A	1896
1884.	Drown, Thomas M	1904
1890.	Gaetzschmann, Moritz	1895
1873.	Gruner, L	1883
1891.	Kerl, Bruno	1905
1895.	Le Conte, Joseph	1901
1891.	Lesley, J. P.	1896
1890.	Patera, Adolph	1890
1886.	Percy, John	1889
1888.	Posepny, Franz	1895
1884.	Richter, Theodor	1898
1899.	Roberts-Austen, W. C.	1902
1890.	Serlo, Albert	1898
1880.	Siemens, C. William	1883
1872.	Thomas, David	1882
1873.	Tunner, Peter R. von	1897
1885.	Wedding, Hermann	1908

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO NOVEMBER, 1910.

No.	Place.	Date.	Trans. Vol.	Trans. Page
1.	Wilkes-Barre, Pa.*	May, '71	1	3
2.	Bethlehem, Pa.	Aug., '71	1	10
3.	Troy, N. Y.	Nov., '71	1	13
4.	Philadelphia, Pa.	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburg, Pa.	Oct., '72	1	25
7.	Boston, Mass.	Feb., '73	1	28
8.	Philadelphia, Pa.*	May, '73	2	3
9.	Easton, Pa.	Oct., '73	2	7
10.	New York, N. Y.	Feb., '74	2	11
11.	St. Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa.	Oct., '74	3	8
13.	New Haven, Conn.	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O.	Oct., '75	4	9
16.	Washington, D. C.	Feb., '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa.	Oct., '76	5	19
19.	New York, N. Y.	Feb., '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y.	Oct., '77	6	10
22.	Philadelphia, Pa.	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y.	Oct., '78	7	103
25.	Baltimore, Md.*	Feb., '79	7	217
26.	Pittsburg, Pa.	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N. Y.*	Feb., '80	8	275
29.	Lake Superior, Mich.	Aug., '80	9	1
30.	Philadelphia, Pa.*	Feb., '81	9	275
31.	Staunton, Va.	May, '81	10	1
32.	Harrisburg, Pa.	Oct., '81	10	119
33.	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo.	Aug., '82	11	1
35.	Boston, Mass.*	Feb., '83	11	217
36.	Roanoke, Va.	June, '83	12	3
37.	Troy, N. Y.	Oct., '83	12	175
38.	Cincinnati, O.*	Feb., '84	12	447
39.	Chicago, Ill.	May, '84	13	1
40.	Philadelphia, Pa.	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn.	May, '85	14	1
43.	Halifax, N. S.	Sept., '85	14	307
44.	Pittsburg, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa.	May, '86	15	lxiii.
46.	St. Louis, Mo.	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb., '87	15	lxxvii.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn.	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala.	May, '88	17	xix.
52.	Buffalo, N. Y.	Oct., '88	17	xxiv.
53.	New York, N. Y.*	Feb., '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct., '89	18	xxiv.
56.	Washington, D. C.*	Feb., '90	18	xxx.
57.	New York, N. Y.	Sept., '90	19	vii.
58.	New York, N. Y.*	Feb., '91	19	xxv.
59.	Cleveland, O.	June, '91	20	xvi.
60.	Glen Summit, Pa.	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburgh, N. Y.	June, '92	21	xxxiii.
63.	Reading, Pa.	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb., '93	21	lii.
65.	Chicago, Ill.	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb., '94	24	xvii.
67.	Bridgeport, Conn.	Oct., '94	24	xxxv.
68.	Florida‡	Mar., '95	25	xix.
69.	Atlanta, Ga.	Oct., '95	25	xxxiii.
70.	Pittsburg, Pa.*	Feb., '96	26	xvii.
71.	Colorado	Sept., '96	26	xxix.
72.	Chicago, Ill.	Feb., '97	27	xvii.
73.	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N. J.*	Feb., '98	28	xvii.
75.	Buffalo, N. Y.	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb., '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix.
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va.*	Feb., '01	31	xix.
81.	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa.§	May, '02	33	xxxv.
83.	New Haven, Conn.	Oct., '02	33	xlvii.
84.	Albany, N. Y.*	Feb., '03	34	xxiii.
85.	New York, N. Y.	Oct., '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xlii.
88.	Washington, D. C.	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa.	Feb., '06	37	xli.
91.	London, England	July, '06	37	xlviii.
92.	New York, N. Y.	April, '07	38	lii.
93.	Toronto, Canada	July, '07	38	lix.
94.	New York, N. Y.	Feb., '08	39	xli.
95.	Chattanooga, Tenn.	Oct., '08	39	xlviii.
96.	New Haven, Conn.	Feb., '09	40	xli.
97.	Spokane, Wash.	Sept., '09	40	xlviii.
98.	Pittsburg, Pa.	Mar., '10	41	xxxviii.
99.	Canal Zone, C. A.	Nov., '10	41	xlv.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ " " " " " " " " " " " " " to Philadelphia.

PUBLICATIONS.

THE publications of the Institute comprise:

TRANSACTIONS.

The volumes of *Transactions*, which are published annually, contain the list of officers, rules, etc., the Proceedings, and the papers revised for final publication. (In this revision, after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there are no important changes.) These volumes are for sale as follows, in paper covers:

Vols. I. to IV. (inclusive), each, ...	$3.00
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Complete set of *Transactions*, Vols. I. to XLI., inclusive, half-morocco binding (freight prepaid), ...	255.00

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Per annum, $10.00. (To members of the Institute, public libraries, educational institutions and technical societies, $5.00.)

Single numbers, $1.00. (To members of the Institute, etc., $0.50.)

Index, Vols. I. to XXXV. (inclusive).—This volume, an octavo of 706 pages, affords a ready and complete reference to any subject treated or alluded to in the *Transactions*, Vols. I. to XXXV., inclusive. The names of persons, mines, works, towns, etc., have been included; and abundant cross-references and classified subheadings have been added to facilitate rapid consultation.

Bound in cloth, $5.00, half-morocco, ... $6.00

Index, Vols. XXXVI. to XL. (inclusive).—This work, containing in rearranged and condensed form the indexes of the *Transactions*, Vols. XXXVI. to XL., inclusive, is of special value in that it supplements the General Alphabetical and Analytical Index of Vols. I. to XXXV., and, by giving all of the new material contained in the technical and professional papers which have been contributed to the Institute during the past five years, brings the index of all the volumes so far published fully up to the date of the last issue, June, 1910. An improvement has been made in the new arrangement of the material under group-headings, which, by presenting the references in tabular form, will enable the reader to find a given item more readily than in the former compact arrangement.

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The Institute maintains at more than a hundred important mining centers throughout the world, free sets of its *Transactions*, open for consultation without fee, to all suitable applicants. Hence, the value of these indexes is by no means limited to individual possessors of complete sets of the *Transactions*. Moreover, the title of a paper, or the record of any remarks concerning a subject, being found in the Index, the Secretary's office of the Institute will supply upon written application any desired information as to the nature and length of said paper, whether it can be supplied in separate pamphlet form, etc.

Special Editions.

"*The Genesis of Ore-Deposits*," comprising the famous treatise of the late Professor Franz Pošepný, with the successive discussions thereof by Le Conte, Blake, Winchell, Church, Emmons, Becker, Cazin, Rickard, and Raymond (all of which were published in Volumes XXIII. and XXIV. of the *Transactions* of the Institute, and subsequently in the special "Pošepný Volume," issued by the Institute); also, later papers by Van Hise, Emmons, Weed, Lindgren, Vogt, Kemp, Blake, Rickard, and others, and the discussions of these papers by De Launay, Beck, and many others (some of these were included in Volume XXX. and the remainder appeared in Volume XXXI.); also a complete bibliography of Institute papers and discussions on this subject from 1871 to 1902.

The original Pošepný volume comprised 265 pages, and was sold for $2.50, at which price the edition was long since exhausted. The present volume is an octavo of 825 pages.

Bound in cloth, $6.00, half-morocco, ... $7.00

"*The Evolution of Mine-Surveying Instruments*." This is a volume of about 400 pages, containing the original paper of Mr. Dunbar D. Scott on that subject (*Transactions*, XXVIII.), first published in

1898, together with later papers, continuing the same subject, and discussions thereof, by Hoskold, Lyman, Davis and many others.

Bound in cloth, ... $3.50

Year Book, containing List of Members, Rules, etc., paper; to Members of the Institute, $0.50; to others, . . 1.00

Glossary of Mining and Metallurgical Terms (1881), cloth, . 1.00

Spanish-American Mining and Metallurgical Glossary, bound in leather, pocket-size, 96 pages, ... 0.75

Chart for the Solution of Kutter's Formula, on cloth, . 0.50

The Conservation of Natural Resources. Papers presented at the Joint Meeting of the four National Engineering Societies, March 24, 1909, by J. R. Freeman, R. W. Raymond, C. W. Baker, and L. B. Stillwell, . 0.25

Pamphlets.

1. The Minutes of the Proceedings of each Meeting.

2. Such of the papers presented or read by title at each Meeting as are furnished by the authors and approved by the Council for full publication. These papers are published separately in pamphlet form, and are marked "subject to revision." Beyond the *Bulletin* edition, a small supply is retained to meet subsequent demand. The stock is nearly complete from 1880. These papers are for sale at the following prices:

No. of Pages.	Single Copies.	10 Copies.	20 Copies.
24 or less	$0.25	$2.00	$3.50
25 to 48	0.30	2.50	4.50
49 to 80	0.40	3.25	5.25
81 to 96	0.45	3.50	6.00
97 to 128	0.50	3.75	6.25
129 to 144	0.55	4.00	6.50
145 to 160	0.60	4.25	6.75
161 to 176	0.65	4.50	7.00

Papers with folders and inserted plates subject to special price.

Authors' Edition of Pamphlets.

Extra copies of pamphlets, if ordered before the printing of the *Bulletin*, will be furnished to members of the Institute at special rates.

Special information concerning all the publications of the Institute will be given on application to Joseph Struthers, Secretary, 29 West 39th St., New York, N. Y.

CONSTITUTION.

[ADOPTED FEB. 21, 1905.]

ARTICLE I.

NAME AND OBJECT.

SEC. 1. This Institute is incorporated under the Membership Corporation Law of the State of New York; its corporate name is AMERICAN INSTITUTE OF MINING ENGINEERS; and its objects are such as are stated in its Certificate of Incorporation.

ARTICLE II.

MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: (1) Members; (2) Honorary Members; (3) Associates; and (4) Honorary Associates. Only Members and Associates residing within the United States of America, Republic of Mexico and Dominion of Canada shall be entitled to vote at the meetings of the Institute.

SEC. 2. All Members, Honorary Members, Associates and Honorary Associates of the American Institute of Mining Engineers as the same existed on the day of the incorporation of this Institute, are Members, Honorary Members, Associates and Honorary Associates, respectively, of this Corporation.

SEC. 3. The following classes of persons shall be eligible for membership in the Institute, namely: as Members and Honorary Members, all professional mining engineers, geologists, metallurgists or chemists, and all persons practically engaged in mining, metallurgy or metallurgical engineering; as Associates and Honorary Associates, all persons desirous of being connected with the Institute who, in the opinion of the Council, are suitable.

SEC. 4. Every candidate for election as a Member or Associate of the Institute must be proposed for election by at least three Members or Associates; must be approved by the Committee on Membership, as prescribed in the By-Laws; and must be elected by the Council. Not less than three-fourths of the votes cast shall be necessary to an election. Every person so elected shall become a Member or Associate, as the case may be, upon payment of his first dues as hereinafter prescribed. Each candidate for Honorary Member or Honorary Associate, must be recommended by at least ten Members or Associates; must be approved by the Council; and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of all the Directors present; provided, however, that the number of Honorary Members and Honorary Associates shall not at any time exceed twenty.

SEC. 5. If any person elected a Member or Associate does not, within sixty days after notice of his election, accept the same and pay his initiation fee and dues for the current year, his election may be cancelled at the discretion of the Council.

SEC. 6. The Council may at any time change the classification of a person elected as an Associate so as to make him a Member, or vice versa. All Members and Associates shall be equally entitled to the privileges of membership, provided that Honorary Members, Honorary Associates, and Members and Associates whose Post-Office addresses shall be outside of the United States, Mexico and Canada, shall not be entitled to vote.

ARTICLE III.

DUES.

SEC. 1. The dues of Members and Associates shall be Ten Dollars per annum, payable in advance on the first day of each Calendar year. Each newly elected Member or Associate shall pay, when notified of election, an initiation fee of Ten Dollars in addition to the dues for the current year. Honorary Members and Honorary Associates shall not be liable to initiation fee or dues. Any Member or Associate in arrears for one year may, at the discretion of the Council, be deprived of the receipt of publications or stricken from the list of Members, provided that he may be restored to membership by the Council on payment of all arrears or may be again proposed and elected after an interval of three years.

SEC. 2. Any Member or Associate not in arrears may become, by the payment of One Hundred and Fifty Dollars at one time, a Life Member or Associate · and shall not be liable thereafter to annual dues.

ARTICLE IV.

BUSINESS MEETINGS OF THE INSTITUTE.

SEC. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall take place on the third Tuesday in February in each year. A report of the financial condition of the Institute and an abstract of the accounts shall be furnished by the Directors, and presented at each annual meeting.

SEC. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint, upon notice to all Members and Associates entitled to vote, directed to each at his last known Post-Office address, and mailed in the City of New York not less than twenty days before the date fixed for such meeting.

SEC. 3. At all business meetings of the Institute the presence of nine Members and Associates shall constitute a quorum.

SEC. 4. At all business meetings of the Institute Members and Associates may vote either in person or by proxy, but no Member or Associate in arrears since the last annual meeting shall be entitled to vote.

ARTICLE V.

OTHER MEETINGS OF THE INSTITUTE.

SEC. 1. All meetings of the Institute other than business meetings shall be held at such times and places as the Council may appoint. Notice of all such meetings shall be given to all Members and Associates by mail.

ARTICLE VI.

Directors and Officers.

Sec. 1. The business and financial affairs of the Institute shall be managed by a Board of Directors, who shall be elected at the annual meeting in the manner prescribed in the Certificate of Incorporation.

Sec. 2. The officers of the corporation shall be a President, Vice-President, Secretary and Treasurer, who shall be elected by the Directors from among their number. All such officers shall be elected at the first meeting of the Board of Directors after each annual meeting of the corporation, and shall hold office for one year or until their successors are elected and qualify.

The duties of all officers shall be such as usually pertain to their offices, respectively, together with such other duties as may from time to time be prescribed for them by the By-Laws. The Treasurer shall give a bond for the faithful performance of his duties in a sum to be fixed by the Board of Directors, but at the expense of the Institute.

Sec. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation or otherwise, the remaining members of the Board may, by a majority vote, elect a successor to fill the vacancy, who shall continue in office until the next annual meeting or until his successor shall have been chosen.

Sec. 4. The Board of Directors may, in its discretion, declare the place of any Director vacant, on his failure for any reason, to attend three successive meetings of the Board. Any Director who shall under this section or in any other manner cease to be a member of the Board shall, at the same time, be held to have vacated any other office to which he shall previously have been elected; and the Board shall elect a new incumbent to the said vacant office.

Sec. 5. The Board of Directors may from time to time appoint from their own number standing and special committees, and may delegate to such committees such duties as they may see fit.

ARTICLE VII.

Meetings of the Board of Directors.

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held on the third Tuesday in February in each year, after the adjournment of the annual meeting of the Institute.

Sec. 2. Special meetings of the Board of Directors, at which any business may be transacted, may be called to meet at any time at the office of the Institute in the City of New York, by notice in writing mailed at least five days before the meeting, by the Secretary to each member of the Board at his last known Post-Office address, signed either by the President or the Vice-President or by three members of the Board.

Sec. 3. At all meetings of the Board of Directors the presence of five members shall constitute a quorum.

ARTICLE VIII.

The Council.

Sec. 1. The professional, technical, scientific and social interests of the Institute shall be committed to the supervision of a Council composed of a President

of the Council, six Vice-Presidents of the Council, a Secretary of the Council and nine Councilors, who shall be elected from among the Members and Associates of the Institute in the manner hereinafter prescribed. Members of the Council may or may not be members of the Board of Directors.

SEC. 2. The President of the Council shall be elected for one year, and no person shall be eligible for immediate re-election to this office who shall have held the same for two consecutive years.

After the first year Vice-Presidents of the Council shall be elected to serve for two years, and Councilors shall be elected to serve for three years. No Vice-President of the Council or Councilor shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. The Secretary of the Council shall be elected annually.

SEC. 3. At the first annual meeting, to be held in the year 1905, there shall be elected a President of the Council to serve for one year, a Secretary of the Council to serve for one year, three Vice-Presidents of the Council to serve for one year, three Vice-Presidents of the Council to serve for two years, three Councilors to serve for one year, three Councilors to serve for two years, and three Councilors to serve for three years. At each subsequent annual meeting there shall be elected a President of the Council to serve for one year; a Secretary of the Council to serve for one year; three Vice-Presidents of the Council to serve for two years; and three Councilors to serve for three years. The term of office of all Members of the Council shall continue until the adjournment of the meeting at which their successors are elected.

SEC. 4. Vacancies in the Council may occur by death or resignation; or the Council may, by the vote of a majority of all its members, declare the place of any officer or member of the Council vacant, on his failure for one year, from inability or otherwise, to attend the regular meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *provided* that the said appointment shall not render such person ineligible for election to the Council at the next meeting.

SEC. 5. The presence of five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or any business coming within the authority of the Council may be transacted at a regularly-called meeting thereof, at which less than a quorum may be present, subject to the approval of a majority of the Council subsequently given in writing to the Secretary and recorded by him with the minutes.

SEC. 6. The election of the Council shall take place at the regular annual meeting of the Institute. Nominations for members of the Council may be sent in writing to the Secretary accompanied with the names of the proposers at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before said meeting, mail to every Member or Associate entitled to vote a list of all nominations for each office so received, together with the names of the persons ineligible for election to each office; and if the Council or a Committee thereof, appointed for the purpose, shall have recommended any nomination, such recommendation may also be sent to the Members and Associates with the list of all nominations made.

ARTICLE IX.

Meetings of the Council.

Sec. 1. Meetings of the Council shall be held at such times and places as the President of the Council or one of the Vice-Presidents of the Council may appoint.

Sec. 2. A meeting of the Council may be held on the day of the annual meeting of the Institute without previous notice. Written notice of all other meetings of the Council, specifying the time and place of such meeting, signed by the Secretary, shall be mailed to every member of the Council at his last known Post-Office address at least ten days before the date of the meeting.

ARTICLE X.

Papers and Publications.

Sec. 1. The Council shall have power to decide as to the acceptance and publication of any professional papers presented to the Institute, subject to such conditions as the Board of Directors may prescribe.

Sec. 2. The copyright of all professional papers communicated to and accepted by the Institute shall be vested in it, unless otherwise expressly agreed between the Council and the author. The Institute shall not assume responsibility for any statements of fact or opinion advanced in the papers or discussions at its meetings. Neither the Council nor the Institute shall officially approve or disapprove any technical or scientific opinion or any proposed enterprise, outside of the management of the meetings, discussions and publications of the Institute, and the conduct of its business affairs by the Board of Directors.

Sec. 3. Special Committees may from time to time be appointed by the Council to make investigations and prepare reports for presentation to the Institute, but no action shall be taken binding the Institute for or against the conclusions embodied in any such reports.

ARTICLE XI.

Suspensions and Expulsions.

Sec. 1. Any member of the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude, shall, upon the passage by the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

Sec. 2. Any member of the Institute may be suspended or expelled for misconduct by the Board of Directors, after charges setting forth such misconduct shall have been prepared by the Council and filed in writing with the Board. Upon the receipt of such charges in writing, the Board may, in its discretion, suspend such member pending a hearing and determination thereupon. As soon as may be after the receipt of such charges, the Board shall fix a date for a hearing thereupon and shall give to the accused member notice thereof in writing, mailed to him at his last known Post-Office address not less than thirty days before said date, accompanied by a full copy of the charges and a copy of the second, third and fourth sections of this article.

Sec. 3. Upon the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any

witnesses who may be called in support of the charges and at his option cross-examine the same; and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defence, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charges. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

SEC. 4. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defence, the Board shall fix a date for a re-hearing within three months from the receipt of such affidavit and shall immediately notify the accused member by mail of such date. Upon the re-hearing, the accused shall have the same privilege of presenting his defence as he would have had upon the original hearing; and after the defence is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

SEC. 5. All interests in the property of the Institute of persons resigning, or otherwise ceasing to be Members or Associates, shall vest in the Institute.

ARTICLE XII.

AMENDMENTS.

SEC. 1. This Constitution or any Article or Section thereof may be amended at any annual meeting by a two-thirds vote of all the members present in person or by proxy, *provided* that notice of the proposed amendment shall have been given in writing at a previous meeting, and *provided also* that the amendment or amendments so adopted shall have been printed and mailed to all Members and Associates not later than thirty days before the annual meeting. Any amendment or amendments approved by a majority of the votes cast shall be deemed to have been adopted, and shall become a part of this Constitution. The Secretary shall forthwith print and distribute to Members and Associates an announcement of the result of said vote, and if any amendment or amendments shall have been adopted, a copy of the section or sections so amended.

BY-LAWS.

[ADOPTED FEB. 21, 1905. AMENDED FEB. 20, 1906, NOV. 16, 1906, AND JAN. 5, 1909.]

I. PRESIDING OFFICERS.

At all Business meetings of the Institute the President, or, in his absence, the Vice-President, or, in the absence of both of them, any other member of the Board of Directors to be chosen by the meeting, shall preside.

At all other meetings of the Institute the President of the Council or, in his absence, one of the Vice-Presidents, if present, shall preside.

II. ORDER OF BUSINESS.

At each Business meeting of the Institute the order of business shall be as follows:

1. Reading of minutes of preceding meeting.
2. Report of the President.
3. Report of the Treasurer.
4. Report of the Secretary.
5. Election of Directors.
6. Election of Members of the Council.
7. Reports of Standing Committees.
8. Reports of Special Committees.
9. Special Orders.
10. Miscellaneous business.

This order of business may be changed by a vote of a majority of the Members and Associates present in person or by proxy.

The usual parliamentary rules shall govern all meetings of the Institute except in cases otherwise provided by the Constitution or the By-Laws.

At all sessions of the Institute other than business meetings, the order of proceedings and the time of adjournment shall rest in the discretion of the presiding officer.

III. SECRETARY.

The Secretary shall keep a record of the proceedings of all meetings of the Institute. He shall be custodian of the Corporate Seal, of the Minute Books, and of all Legal Documents belonging to the Institute. He shall conduct, on behalf of the Institute, all correspondence relating to business matters, except such as pertains directly to the office of the Treasurer.

He shall notify all officers and Directors and Members of the Council, and all Members of Committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other meetings of the Institute; and shall, in calling special meetings of the Directors, specify the object of such meeting.

IV. SECRETARY OF THE COUNCIL.

The Secretary of the Council shall act as the Clerk of that body at all of its meetings and at all meetings of the Institute called for the discussion of professional, technical or scientific matters, or for any other purpose than the transaction of business.

He shall be custodian of all technical or scientific papers submitted to the In-

stitute for its consideration, shall have charge of the editing and printing of all material published by the Institute, and of the distribution thereof. On the first day of May following the year in which each volume of *Transactions* is printed, he shall turn over to the Library Committee all copies of the same not theretofore distributed by him. He shall have charge of all the correspondence of the Institute relating to other than business affairs.

The Secretary of the Council shall receive a salary to be fixed by the Board of Directors. He may appoint an Assistant with the title of Editor, who shall likewise receive a salary to be fixed by the Board of Directors.

The Secretary of the Council may or may not be the same person as the Secretary of the Institute.

V. Assistant Secretary.

The Secretary may, with the approval of the Board of Directors, appoint an Assistant to whom both he and the Secretary of the Council may delegate such of his or their duties as he or they may see fit. This Assistant Secretary shall receive such salary as shall be fixed by the Board of Directors, which shall cover his services both to the Secretary and to the Secretary of the Council.

VI. Treasurer.

The Treasurer shall collect and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute and at every meeting of the Board of Directors at which such report shall be called for, the balance of money on hand, and any existing appropriation which may affect the same.

His accounts shall be audited annually by a Committee of three Members or Associates to be appointed by the President at least thirty days prior to the annual meeting in each year, which Committee shall report thereon at such annual meeting.

The Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding $5,000, in a special account in a Bank or Trust Company, subject to the draft of the Assistant Treasurer, and may delegate to the Assistant Treasurer the duty of paying, out of this account, the current expenses of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to the Assistant Treasurer or not.

VII. Assistant Treasurer.

The Treasurer may appoint, with the approval of the Board of Directors, an Assistant Treasurer, to whom he may delegate the duty of conducting the correspondence incidental to the office of Treasurer, of receiving and depositing in bank to the credit of the Institute all moneys received, and of paying, out of the special account upon which he may be authorized to draw, the necessary expenses of the Institute. The Treasurer may require of him a bond, running to the Treasurer personally, in an amount not exceeding $5,000, the expense of which shall be borne by the Institute.

The Assistant Treasurer shall receive such compensation as shall be fixed by the Board of Directors.

The offices of the Assistant Secretary and of the Assistant Treasurer may, if

so desired by both the Secretary and the Treasurer and approved by the Board of Directors, be united in the same person, who shall then receive the salary of both offices.

The Assistant Treasurer may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical and office force for himself, the Assistant Secretary and the Secretary of the Council, at such salaries as shall be approved by the Board of Directors. He shall, if the offices of Assistant Secretary and Assistant Treasurer be united in the same person, be the immediate superior of all such employees, unless the Secretary of the Council or the Treasurer be present, in which event either of them shall be the superior of all employees, including their respective assistants.

VIII. Standing Committees.

The Standing Committees of the Institute shall be three in number, known respectively as the Finance Committee, the Library Committee and the Committee on Membership.

The Finance Committee and the Library Committee shall each consist of three members of the Board of Directors, and shall be appointed by the President at the first meeting of the Board, after the annual meeting in each year.

The Committee on Membership shall consist of five Members of the Council, and shall be appointed by the President of the Council, at the first meeting of the Council after the first annual meeting in each year.

IX. Finance Committee.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute, and to consider ways and means of increasing its revenues and of limiting its expenses. It shall report from time to time to the Board as often as it may deem expedient, and whenever it shall be directed so to do ; and the Treasurer shall at all times furnish it with such statements and information as it may desire.

It shall determine the investment of such surplus moneys as shall from time to time accrue to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute in the custody of the Treasurer, and report thereon to the Board.

It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer.

The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested to do so.

X. Library Committee.

The Library Committee shall be the custodian of all books in the Institute Library and of additions thereto ; also of all back numbers of the *Transactions* of the Institute. It shall, on the first day of May, of each year, receive from the Secretary of the Council, and receipt for same to him, all the volumes of *Transactions* for the preceding year, not then distributed by said Secretary.

It shall cause to be kept, under the direction of the Assistant Secretary, a catalogue of all books in the Library and an account in ledger form of all volumes of *Transactions* in its custody, in which shall be charged to it all volumes delivered to it, and in which shall be credited all volumes taken from its custody for sale or for any other purpose.

The receipts from the sale of any volume of *Transactions* taken from the custody of the LIBRARY COMMITTEE shall be credited to the LIBRARY COMMITTEE on the books of the Treasurer, and devoted to the general purposes of the Institute.

XI. COMMITTEE ON MEMBERSHIP.

All nominations for Members or Associates of the Institute shall be submitted to and passed upon by the COMMITTEE ON MEMBERSHIP, who shall report thereon to the Council. It shall receive and consider all communications respecting candidates, and shall make diligent inquiry as to the character and qualifications of each one. Its proceedings shall be secret and confidential.

No member of the Committee shall propose any candidate.

XII. ELECTION OF MEMBERS.

After the COMMITTEE ON MEMBERSHIP shall have reported to the Council its conclusions as to the acceptability of each candidate, the Council shall vote upon the same.

Two negative votes of members of the Council present shall prevent the election of any candidate. No person shall be proposed for election to the Institute within one year after his name shall have been rejected by the Council.

XIII. UNITED ENGINEERING SOCIETY.

The Board of Directors shall, at its first meeting after the adoption of these By-Laws, designate three Members or Associates of this Institute to be representatives of this Institute upon the Board of Trustees of the UNITED ENGINEERING SOCIETY, making at the same time provision for the expiration of the terms of office of said representatives, as provided in the By-Laws of the said UNITED ENGINEERING SOCIETY.

At the last meeting of the Board of Directors prior to the first day of each January thereafter, the Board shall designate a Member or Associate of this Institute to be a representative of this Institute upon the Board of Trustees of the said UNITED ENGINEERING SOCIETY for a period of three years beginning at the next ensuing annual meeting of said Society.

At any time when a vacancy shall occur in the representation of this Institute in the Board of Trustees of said Society, by reason of the death, resignation or removal of any such representative therein, the Board of Directors of this Institute shall designate a Member or Associate to fill such unexpired term.

XIV. PUBLICATIONS.

The publications of the Institute shall include a periodical, called the *Bulletin* of the American Institute of Mining Engineers, which shall contain reports of proceedings, professional papers, notices, and other matter of interest to members. From the annual dues paid by each Member or Associate, five dollars shall be deducted and applied as a subscription to the *Bulletin* for the year covered by such payment.

XV. AMENDMENTS.

These By-Laws may at any time be altered or amended by a vote of two-thirds of the Board of Directors, or by the Members, at a business meeting of the Institute, in the same manner provided for amendments of the Constitution in Article XII. thereof.

ANNUAL MEETING OF THE INSTITUTE.

At the Annual Business Meeting of the Institute, held Feb. 15, 1910, the following officers were elected:

COUNCIL.

President.

(To serve for one year.)

D. W. BRUNTON, ... Denver, Colo.

Vice-Presidents.

(To serve for two years.)

BENJAMIN B. LAWRENCE, ... New York, N. Y.
JOSEPH W. RICHARDS, ... South Bethlehem, Pa.
ALBERT SAUVEUR, - - . . . . Cambridge, Mass.

Secretary.

(To serve for one year.)

R. W. RAYMOND, ... New York, N. Y.

Councilors.

(To serve for three years.)

ROBERT E. JENNINGS, ... New York, N. Y.
WILLIAM KELLY, ... Vulcan, Mich.
CHARLES F. RAND, ... New York, N. Y.

DIRECTORS.

(To serve for three years.)

JAMES GAYLEY, ... New York, N. Y.
CHARLES KIRCHHOFF, ... New York, N. Y.
FRANK LYMAN, ... New York, N. Y.

[SECRETARY'S NOTE.—The complete list of all officers of the Institute will be found on p. v. of this volume. The following explanation, first published in *Bi-Monthly Bulletin*, No. 8, March, 1906, p. viii., is here repeated in order to recall to old members, and convey to new ones, the relations of the two governing bodies as determined by the Certificate of Incorporation of the Institute, and the Constitution and By-Laws adopted in accordance therewith.

The body legally responsible for the business management is the Board of nine Directors (three elected annually to serve three years), which elects its own officers. This body, for reasons of practical convenience, is composed of well-known members residing in New York City, and able to attend, without serious inconvenience or expense, the necessary meetings of the Board. The officers of this Board are legally the officers of the Institute. But, apart from business management, the Board exercises no control over the election of members, or the professional and technical work of the Institute, except that its vote is required to elect honorary members, upon the recommendation of the Council.

The Council is a body constituted in all respects (except that it has no Treas-

urer) like the Council existing before the incorporation of the Institute, in January, 1905, and charged with all duties and powers, except those which the Board of Directors must legally perform. It elects members, appoints the times and places of professional meetings, and controls the publication and distribution of papers and volumes, etc. Its members (President, Vice-Presidents and Councilors) are elected by the members of the Institute, voting in person or by proxy, and after publication of the nominations received; and it is intended to represent, as far as practicable, both the professional and the geographical distribution of the membership. Consequently, whatever professional honor attaches to official position belongs to membership in the Council, rather than in the legal Board of Directors. This remark implies no disparagement of the members of the latter body, every one of whom has served, or is now serving, as a member of the Council. But it is only fair to explain that their election and continued re-election as Directors is simply a matter of legal convenience.]

Proposed Amendments to the Constitution.

The consideration of the proposed amendment to Article III. of the Constitution, whereby, in the first line of said article, the word "fifteen" should be substituted for the word "ten," was postponed to an adjourned session of the annual meeting, 1910, to be held at a time fixed by the Board of Directors upon due notice given as provided by Article IV., Section 2, of the Constitution for the calling of special business meetings.

Provided that, if the Board of Directors shall not call such adjourned session, then the consideration of this proposed amendment shall be in order at the Annual Meeting of February, 1911, and

Provided also that, for the purpose of the said consideration of this amendment, or any proposed amendment thereof, new proxies shall be sent to all members and associates entitled to vote, which shall be so drawn as to cover the questions thus concerned, and permit an intelligent vote thereon.

PROCEEDINGS OF THE BOARD OF DIRECTORS.

The following acts of the Directors are reported for the information of members:

At a meeting held Nov. 11, 1909, after unanimous recommendation by the Council, M. Alexandre Pourcel was unanimously elected an honorary member of the Institute in recognition of his distinguished researches in the science and contributions to the practice of the metallurgy of iron and steel.

The Secretary reported that books, etc., were from time to time received as gifts by the United Engineering Society, and therefore could not be properly treated as belonging to the library of either of the three Founder Societies, although the said societies were, in some sense, owners thereof through their connection with the United Engineering Society. Also, that the Library Conference Committee, consisting of one representative from the Library Committee of each of the three societies, desired, from time to time, the purchase of reference-books, such as directories, dictionaries, etc., or of library equipment for the joint use of the societies, and that articles so purchased could not properly be carried on the inventory of the library of either society.

It was voted: (1) That gifts of books, etc., received by the United Engineering Society, shall be regarded as the property of that society; and the Library Committee of the Institute is hereby discharged, concerning such books, etc., from the responsibility imposed by By-Law X.

(2) That the Secretary is hereby authorized to consent to the purchase, for joint use of the Founder Societies, of such books or library equipment as may be approved by the unanimous vote of the Library Conference Committee, and to pay from the funds of the Institute its proportion of one-third the cost thereof. *Provided*, That said payments, together with the purchases of books, etc., for the library of the Institute, shall not exceed the amounts appropriated by this Board for library purchases and bindings.

At a meeting held Jan. 10, 1910, Mr. Theodore Dwight, Treasurer of the Land Fund Committee, presented a report of

the work done by this committee during the year 1909, which shows a total of subscriptions collected during the year of $22,589.75, and promised subscriptions of $17,600. Payments were made by this committee on the land mortgage during the year amounting to $22,000, which, together with the $15,000 paid by the Institute, reduces the debt of the land fund from $125,000, Jan. 1, 1909, to $88,000, Jan. 1, 1910. The deferred payments, amounting to $17,600, together with the present balance on hand of $1,139.75, totaling $18,739.75, will further reduce the balance due on the land mortgage to $69,260.25.

Dr. Joseph Struthers was unanimously elected a trustee of the United Engineering Society, to serve for a term of three years, succeeding Mr. Theodore Dwight.

At a meeting held Feb. 15, 1910, the following officers were elected: *President*, James Gayley; *Vice-President*, James Douglas; *Secretary*, R. W. Raymond; *Treasurer*, Frank Lyman.

The following standing committees were appointed to serve during the ensuing year:

Finance Committee: James Douglas, Theodore Dwight, and Albert R. Ledoux.

Library Committee: James F. Kemp, Charles H. Snow, and R. W. Raymond.

Financial Statement.

The following statement of receipts and expenditures from Jan. 1 to Dec. 31, 1909, is published by authority of the Board of Directors:

RECEIPTS.

Balance from statement of January, 1909,		$3,888.58
Annual dues,*	$37,909.18	
Life memberships,	890.00	
Initiation fees,	1,990.00	
Binding of *Transactions*,	3,668.30	
Sale of publications, electrotypes, advertising, and miscellaneous receipts,	9,768.77	
		54,226.25
Interest on bonds and deposits,		184.66
Sale of 6 $1,000 N. Y. Central & Hudson River R.R. Lake Shore Collateral 3½ per cent. Coupon Bonds (net),	$5,140.83	
Sale of 13 $1,000 Pennsylvania R.R. Co. 6 per cent. Registered Bonds (net),	13,567.93	
		18,708.76
		$77,088.25

* $18,875 of this amount has been applied to subscriptions to the *Bulletin* in accordance with post-office regulations.

DISBURSEMENTS.

Printing Vol. XXXIX. of the *Transactions*, *Bulletin*, extra pamphlets, and advertising expenses, etc.,	$12,355.34	
Printing circulars and ballots,	231.10	
Binding Vol. XXXIX. of the *Transactions*,	3,458.00	
Binding miscellaneous volumes,	465.55	
Engraving and electrotyping,	567.51	
Secretary's department, including clerks, stenographers, and expenses of editing and proof-reading, and special assistance in connection with meetings,	11,428.60	
Treasurer's department, including collection of dues, shipping, etc.,	6,087.50	
Librarian and assistants,	1,854.80	
Postage,	3,711.33	
Stationery,	554.31	
Express and freight charges,	1,792.10	
Telephone,	207.90	
Telegrams, cables, carfares, etc.,	57.96	
Office supplies and repairs,	76.04	
Refunding miscellaneous payments,	138.22	
Insurance premiums (Surety),	95.00	
Collection charges,	37.16	
Extra clerical assistance,	76.66	
Special stenographers and expenses of meetings,	1,722.55	
Auditing,	125.00	
Office cleaning and sundry expenses,	32.55	
		$45,075.18
Principal on account, Land Mortgage,	15,000.00	
Interest at 4 per cent., for 1909, on unpaid balance of land mortgage on 25 to 33 West 39th St. ($125,000, January 1, 1909, reduced to $88,000 January 1, 1910),	4,276.66	
Quota of current expenses of building 25 to 33 West 39th St.,	6,000.00	
		25,276.66
Special editing, part payments on printing and binding special edition, new volume of Genesis of Ore-Deposits,		39.00
Library equipment,		173.33
Library additions of books, periodicals, etc., binding of exchanges, and stationery (expenditure from appropriation of $1,000),		827.04
Furniture and fixtures,		68.45
Balance,		5,548.59
		$77,008.25

NEW YORK, N. Y., February 7, 1910.

We have examined the above statement, compared it with the books and vouchers and find same correct.

(Signed) BARROW, WADE, GUTHRIE & Co.,
Certified Public Accountants.

REPORT OF THE COUNCIL FOR THE YEAR 1909.

The following report is published for the information of members:

Meetings.

There were two meetings of the Institute held during the year 1909 for the reading and discussion of papers—the Ninety-sixth Meeting, in New Haven, February 23 to 25, and the Ninety-seventh Meeting, in Spokane, September 28 to 29.

A detailed record of the proceedings of these meetings, including a description of the entertainments and excursions connected therewith, has been published and duly distributed to the members: the New Haven meeting in *Bulletin* No. 28, April, 1909, pp. 425 to 437, and the Spokane meeting in *Bulletin* No. 36, December, 1909, pp. 1059 to 1118. At the New Haven meeting there were presented 27 papers and 5 discussions, oral and written; in these discussions 18 separate contributors participated. At the Spokane meeting there were presented 27 papers and 8 discussions, oral or written; in these discussions 14 contributors participated. At the New Haven meeting the names of 145 members and guests were registered at the Institute headquarters; this number, however, does not represent all who were present at the sessions and the excursions to Ansonia and Bridgeport. In connection with the Spokane meeting, the number of members and guests registered was approximately 200, but the total number participating in the trip through the Yellowstone Park and the excursions to Butte, Anaconda, and the Cœur d'Alène district, or attending, in whole or in part, the sessions at Spokane, and the subsequent excursions to Seattle, Tacoma, Portland and the Dalles of the Columbia, Salt Lake City, Bingham Cañon, Glenwood Springs, and through the Royal Gorge to Pueblo and Colorado Springs, exceeded 400.

Publications.

Transactions.—Volume XXXIX. of the *Transactions*, an octavo of 1,006 pages, comprising 50 papers and 16 discussions presented during the year 1908, was issued and distributed to

members in June, a few weeks earlier in the year than the corresponding appearance of Volume XXXVIII. With the exception of the index, now being compiled, the material for Volume XL., forming in all about 1,000 pages, is in the hands of the printer, and it is expected that the bound volume will be off the press and ready for distribution in June, 1910.

Bulletin.—Twelve numbers of the *Bulletin* (Nos. 25 to 36), containing the technical papers and discussions of the Institute (in "subject to revision" form) and announcements of general interest to the members of the Institute, such as library accessions and requirements during the year 1909; notices of meetings of the Institute and of other societies; lists of proposed members and associates; changes of address; deaths of members; obituary notices; Index of Subjects and Authors, etc., have been published and distributed promptly throughout the year 1909. The number of pages occupied by technical papers and discussions amounts to 1,210, to which are to be added 326 pages of announcements, and 184 pages of advertising matter, making a total of 1,720 pages of printed matter.

The editorial and business management of the *Bulletin*, Volume XXXIX. and the forthcoming Volume XL. of the *Transactions* continues in charge of Dr. Joseph Struthers, Assistant Secretary and Editor of the Institute.

Membership.

Changes in membership have taken place during the year as follows: 1 honorary member, 1 honorary associate, 190 members and 11 associates have been elected; 6 members have been reinstated, 4 members re-elected, 2 members have become honorary members, and 6 associates have become members; the deaths of 32 members and 1 associate have been reported; 60 members and 4 associates have resigned; and 67 members and 6 associates have been dropped from the roll by reason of non-payment of dues, loss of correct address, etc.* These changes are shown in the accompanying table.

The total membership on Jan. 1, 1910, was 4,284, as compared with 4,241 on Jan. 1, 1909—a net gain for the year of 43 members.

* Many of these, no doubt, will be reinstated, as has been the case in former years.

Membership of the American Institute of Mining Engineers, Jan. 1, 1910.

	Honorary Members.	Honorary Associates.	Members.	Associates.	Totals
Membership Dec. 31, 1908	11	....	4,066	164	4,241
Gains: By Election	1	1	190	11	203
Change of Status	2	....	6	....	8
Reinstatement	....	....	6	....	6
Re-election	....	....	4	....	4
Losses: By Resignation	....	....	60	4	64
Change of Status	....	....	2	6	8
Dropping	....	....	67	6	73
Death	....	....	32	1	33
Total gains	3	1	206	11	221
Total losses	....	....	161	17	178
Membership Dec. 31, 1909	14	1	4,111	158	4,284

The list of deaths reported during the year 1909 comprises the following names, the figures in parentheses indicating the year in which the persons named were elected to membership.

Members and Associates.—William Adams (1905), Robert S. Brooks (1906), James W. Brown (1905), Raymond B. Brown (1895), F. G. Bulkley (1882), Dr. Charles B. Dudley (1878), Richard Eames, Jr. (1888), George G. Francis (1880), Persifor Frazer (1871), Edward L. Fuller (1904), John M. Grice (1908), Rasmus Hanson (1896), Harold H. Harvey (1903), Alphonse Hennin (1888), H. August Hunicke (1883), Algernon K. Johnston (1899), Jerome Keeley (1876), Emil Krabler (1891), L. G. Laureau (1878), Frank A. Lucy (1903), John E. McCurdy (1897), Charles C. Mattes (1881), William Metcalf (1875), Stephen C. Miller (1904), Israel W. Morris (1875), John W. Nesmith (1896), Bertel Peterson (1900), Robert Pitcairn (1881), Jasper R. Rand (1900), Davis Richardson (1905), William H. Singer (1873), James Stirling (1906), Delos V. A. Williams (1903), T. F. Witherbee (1871).

MEMBERSHIP.

The following list comprises the names of those persons elected as members, who duly accepted election during the year 1910. The marks used to designate the different classes of membership are: Life Member, **; Member,*; Associate Member, †. Heavy-faced type signifies Honorary Membership.

*John A. Agnew, ... Kalgoorlie, West. Australia.
*Richard E. Armstrong, ... Bingham Canyon, Utah.
*John F. Austin, ... Monterey, Mexico.
*Robert L. Bailie, ... Detroit, Mich.
*Frank H. Blackmar, ... San Martin de Loba, Colombia, So. Am.
*Alfred W. Bleeck, ... Rangoon, Burma, India.
*Edwin S. Boalich, ... Searchlight, Nev.
*John W. Boileau, ... Pittsburg, Pa.
*Harry M. Booth, ... Denver, Colo.
*Homer F. Braddock, ... Greensburg, Pa.
*Cuthbert H. Brailsford, ... Sheffield, England.
*Nelson G. Brayer, ... Sharon, Pa.
*Gilmour E. Brown, ... Applegarth, Ballock, Scotland.
*Lowell H. Brown, ... Catasauqua, Pa.
*Frederick J. Brulé, ... Tooele, Utah.
*Christopher Bruns, Jr., ... Cumanayagua, Santa Clara, Cuba.
*John E. Butler, ... Stearns, Ky.
*Mel C. Butler, ... Fairfax, Wash.
*Reginald H. B. Butler, ... London, England.
*Halsted W. Caldwell, ... Thomas, Ala.
*Dean S. Calland, ... Pachuca, Hid., Mexico.
*Robert B. Carnahan, Jr., ... Middletown, Ohio.
*Edwin E. Carpenter, ... Syracuse, N. Y.
*William R. Chedsey, ... Moscow, Idaho.
*William W. Charles, ... Tonapah, Nev.
*Marco Chiapponi, ... New York, N. Y.
*Arthur O. Christensen, ... Sombrerete, Zac., Mexico.
*Fritz Cirkel, ... Montreal, Canada.
*Frederick G. Clapp, ... Pittsburg, Pa.
*Camille Clerc, ... Paris, France.
*James B. Cook, ... Punxsutawney, Pa.
*Maurice D. Cooper, ... Buffalo, N. Y.
*Edward H. Coxe, ... Birmingham, Ala.
*Rufus C. Crawford, ... Pittsburg, Pa.
*Herbert H. Crease, ... Santana, Colombia, So. Am.
*Andrew B. Crichton, ... Johnstown, Pa.
*William N. Cummins, ... Worth, W. Va.
*Melvin P. Dalton, ... Poza, Son., Mexico.
*John Daniell, ... Laurium, Mich.

*Harold P. Davis, ... New York, N. Y.
*William J. Davis, Jr., ... San Francisco, Cal.
*Alpheus E. Deardorff, ... Unsan Mines, Korea.
*Justin S. De Lury, ... Moscow, Idaho.
*Archibald F. Dick-Cleland, ... El Oro, Mex., Mexico.
*Carl F. Dietz, ... Boston, Mass.
*James S. Douglas, ... Douglas, Ariz.
*Byron E. Eldred, ... New York, N. Y.
*Richard Eustice, ... Moonta, So. Australia.
*Harry E. Ewing, ... Worth, W. Va.
*Milton H. Fies, ... Birmingham, Ala.
*David M. Folsom, ... Stanford University, Cal.
*Heitaro Fujita, ... Dojima, Kita-ku, Osaka, Japan.
*John P. Furbeck, ... Oak Park, Ill.
*Malcolm M. Galbraith, ... Amboy, Cal.
*Archibald D. Gilchrist, ... Ballarat, Vic., Australia.
*Thomas W. Graham, ... Workington, England.
*George N. de Gripari, ... Salonica, Turkey.
*Benjamin F. C. Haanel, ... Ottawa, Ont., Canada.
*George T. Haldeman, ... Pittsburg, Pa.
*Edmund C. Harder, ... Washington, D. C.
*Thomas S. Harrison, ... Wiley, Wyo.
*William H. Hayes, ... Moonta Mines, So. Australia.
*William W. Hearne, ... Philadelphia, Pa.
*Thomas F. Higgins, ... Panulcillo, Coquimbo, Chile, So. Am.
*Rudolf Hoffmann, ... Clausthal, Germany.
*Charles Holding, ... Collinsville, Ill.
*Charles B. Hollis, ... Randolph, Vt.
*Edwin O. Holter, ... New York, N. Y.
*Walter Hooker, ... London, England.
*Andrew M. Howat, ... Los Angeles, Cal.
*Frank U. Humbert, ... Santiago de Cuba, Cuba.
*James M. Hyde, ... Palo Alto, Cal.
*Wallace G. Imhoff, ... Pittsburg, Pa.
*J. F. Inglis, ... Salt Lake City, Utah.
*Joseph H. Ivey, ... Quechisla, Bolivia, So. Am.
*Harry E. Judd, ... Sakchi, B. N. R., India.
*Benzo Katsura, ... New York, N. Y.
*Takeshi Kawamura, ... Akita, Japan.
*Lynn Y. Keady, ... Portland, Ore.
*Dyke V. Keedy, ... Boston, Mass.
*Kazuye Kibe, ... Nikko, Japan.
*Richard Kleesattel, ... Seattle, Wash.
*Junjiro Kobuse, ... Kosaka, Akita-Ken, Japan.
*Jules Labarthe, ... Salt Lake City, Utah.
*David H. Ladd, ... Wallaroo, So. Australia.
*Frank C. Laurie, ... Mexico City, Mexico.
*Nathan O. Lawton, ... Miami, Ariz.
*Montrose L. Lee, ... Oxford Furnace, N. J.
*David Lemmon, ... Pioche, Nev.
*Grenville Lewis, ... Pineville, Ky.
*David S. Longacre, ... Dragon, Utah.

*Joseph B. S. McIntosh, ... Tooele, Utah.
†William E. Manning, ... Youngstown, Ohio.
*Stuart B. Marshall, ... Dunbar, Pa.
*Harry B. Meller, ... Pittsburg, Pa.
*Pomeroy C. Merrill, ... Hibbing, Minn.
*Arthur J. Miller, ... Salt Lake City, Utah.
*Walter C. Minsch, ... Silverton, Colo.
*Charles Mitke, ... Dawson, N. M.
*Bertie H. Moore, ... Kalgoorlie, West. Australia.
*Everett B. Moore, ... Fairmont, W. Va.
*Howard W. Morgan, ... Kirkland, Ariz.
*Frank G. Morris, ... Sayreton, Ala.
*Walter L. Morrison, ... Brooklyn, N. Y.
*William Neilson, ... Midas, Nev.
*J. Wilfrid Newbery, ... Kalgoorlie, West. Australia.
*Peter A. Newton, ... So. Chicago, Ill.
*John Orr, ... Johannesburg, So. Africa.
*Philip B. Osborn, ... Johannesburg, So. Africa.
*Charles G. Osborne, ... So. Chicago, Ill.
*Hugh D. Pallister, ... Cleveland, Ohio.
*Karl C. Parrish, ... Des Moines, Iowa.
*Robert L. Parrish, ... Covington, Va.
*Benjamin G. Patterson, ... Mount Morgan, Queensland, Australia.
*John E. Penberthy, ... Globe, Ariz.
*Edwin T. Perkins, ... Granby, Mo.
*Walter I. Phillips, ... Cheyenne, Wyo.
*Charles E. Phoenix, ... Bellingham, Wash.
†John S. Piper, ... Lawrenceville, Ill.
*H. Robinson Plate, ... New York, N. Y.
*Albert F. Plock, ... Mayville, Wis.
*James H. Polhemus, ... Carterville, Mo.
*Theodore H. Proske, ... Denver, Colo.
*Edward W. Ralph, ... Kimberly, Nev.
*William C. J. Rambo, ... Cainesville, Mo.
*Francis C. Richard, ... Carlton, Vic., Australia.
*Joseph W. Roe, ... New Haven, Conn.
*Ludwig Rose, ... Berlin, Germany.
*William P. Rutherford, Jr., ... Glasgow, Scotland.
*W. A. Ryan, ... Inyati, Rhodesia, So. Africa.
*Irving L. Ryder, ... San José, Cal.
*Roswell E. Sampson, ... Medford, Mass.
*Charles C. Selbie, ... So. Pasadena, Cal.
*Oscar Simmersbach, ... Breslau, Germany.
*Alexander Sharp, ... Orient, Wash.
*Charles H. Smith, ... Baltimore, Md.
*Lyon Smith, ... Cortez, Nev.
*Hedley V. Smythe, ... Bay of Islands, Newfoundland.
†Theodore Sternfeld, ... New York, N. Y.
*Maxwell Stevenson, Jr., ... Philadelphia, Pa.
*Charles A. Stewart, ... Ithaca, N. Y.
*Robert B. Stewart, ... Toronto, Ont., Canada.
*Alfred W. Stickney, ... Cambridge, Mass.

*Isuzu Sugimoto, ... Tokyo, Japan.
*Roy A. Sulliger, ... Los Angeles, Cal.
*Arthur M. Swartley, ... Denver, Colo.
*Whitman Symmes, ... Virginia City, Nev.
*Stephen Taber, ... University, Va.
*Thomas K. Taft, ... Telluride, Colo.
*James Thame, ... Sheen, Surrey, England.
*Harry S. Thayer, ... Colorado Springs, Colo.
*Bruno Thomas, ... Seattle, Wash.
*Malcolm MacF. Thompson, ... Ridgefield, N. J.
*Frank L. Torres, ... Monterey, N. L., Mexico.
*Kennosuke Tsujimoto, ... Dojima, Osaka, Japan.
*Umenoshin Tsukakoshi, ... Douglas, Ariz.
*Adin P. Tyler, ... Juneau, Alaska.
*John Tyssowski, ... Washington, D. C.
*Rupert F. Vaughan, ... Celebes, Ned. India.
Tsunashiro Wada, ... Ishigaya, Tokyo, Japan.
*W. Rogers Wade, ... Tyrone, N. M.
*Thor Warner, ... Haileybury, Ont., Canada.
*Frank M. Warren, ... Minneapolis, Minn.
*Ralph W. Watson, ... Callao, Utah.
*Karl M. Way, ... Boston, Mass.
*William P. Wescott, ... Jersey City, N. J.
*Lewis A. Westcott, ... Many Peaks, Queensland, Australia.
*John D. Whitney, ... Elk City, Idaho.
*Leslie J. Wilmoth, ... Germiston, So. Africa.
*Charles S. Witherell, ... Newark, N. J.
*Hugh M. Wolflin, ... Seattle, Wash.
*Robert B. Woodworth, ... Pittsburg, Pa.
*Charles O. Wright, ... Saxman, W. Va.
*Kisaburo Yamaguchi, ... Nikko, Japan.
*Arthur Yates, ... Lebong Soelit, Sumatra, D. E. I.
†Carl D. Young, ... Rolla, Mo.

Deaths.

The following list comprises the names of members whose deaths have been reported to the Secretary of the Institute during the year 1910:

Date of Election.	Name.	Date of Decease.
1905.	*Masayoshi Abe,	November 12, 1909.
1903.	*W. Edward Adams,	December 13, 1910.
1887.	*James Archbald,	October 5, 1910.
1890.	*John H. Bartlett,	July 14, 1910.
1881.	*William F. Biddle,	August 10, 1910.
1871.	*William P. Blake,	May 22, 1910.
1902.	*Wager Bradford,	July 9, 1909.
1887.	*Arthur Brock,	December 23, 1909
1895.	†Fayette Brown,	January 20, 1910.
1900.	*Henry Burrell,	April 2, 1910.

Date of Election.	Name.	Date of Decease.
1901.	*José Calero,	———, 1910.
1899.	*Frank J. Campbell,	March 6, 1910.
1887.	*Frank R. Carpenter,	April 1, 1910.
1879.	*Octave Chanute,	November 23, 1910.
1890.	*Edmund W. Coddington,	June 9, 1910.
1903.	*R. Prewitt Coleman,	February 5, 1910.
1907.	*Francis V. Drake,	———, 1909.
1891.	*Charles H. Ferry,	May 2, 1910.
1894.	*James W. Fuller,	January 15, 1910.
1879.	*Paul A. Fusz,	February 16, 1910.
1881	*Edward C. Hegeler,	June 4, 1910.
1886.	*Gus C. Henning,	December 30, 1910.
1884.	*A. D. Hodges, Jr.,	November 7, 1910.
1876.	*John W. Hoffman,	January 18, 1910.
1884.	*Ottokar Hofmann,	December 24, 1909.
1906.	*Thomas A. Irvin,	January 2, 1910.
1889.	*Guy R. Johnson,	June 22, 1910.
1907.	*Alfred Kimber,	April 7, 1910.
1905.	*Herbert H. Light,	December 28, 1909.
1902.	*Edmund D. North,	June 23, 1910.
1900.	*Josiah Owen,	December 19, 1909.
1874.	*Charles B. Parsons,	January 28, 1910.
1906.	*Ernest Y. Pomeroy,	September —, 1909.
1905.	*Pietro Redaelli,	April 10, 1910.
1900.	*Ferd H. Regel,	January 18, 1910.
1888.	*William H. Schlemm,	———, 1910.
1906.	*John C. Sevier,	November 11, 1909.
1899.	*H. A. Shipman,	January 2, 1910.
1881.	*Albert Spies,	August 16, 1910.
1897.	*Herbert S. Stark,	February 18, 1910.
1887.	*John Sutcliffe,	July 23, 1910.
1897.	*James P. Wallace,	June 19, 1910.
1900.	**Thomas F. Walsh,	April 8, 1910.
1892.	*S. Bowman Wheeler,	July —, 1909.
1907.	*Wilfred F. Wheeler,	November 18, 1909.
1899.	**Frederick de L. Williams,	August 22, 1909.
1882.	*Alvinus B. Wood,	January 24, 1910.
1900.	*Alfred F. Wuensch,	January 21, 1910.

* Member. ** Life Member. † Associate.

Proceedings of the Ninety-Eighth Meeting, Pittsburg, Pa., March, 1910.

COMMITTEES.

LOCAL COMMITTEE.—R. C. Crawford, *Chairman;* Harrison W. Craver, *Secretary;* Julian Kennedy, Taylor Allderdice, E. W. Pargny, Charles L. Miller, W. H. Rea, S. A. Taylor, M. E. Wadsworth, W. M. Henderson, A. C. Dinkey, W. L. Jones, E. L. Messler, George Mesta, J. H. Jones, G. H. Riley, S. L. Goodale, D. W. McNaugher.

LADIES' COMMITTEE.—Mrs. Taylor Allderdice, *Chairman;* Mrs. Julian Kennedy, Mrs. R. C. Crawford, Mrs. E. M. Herr, Mrs. J. W. Marsh, Mrs. T. A. Mellon, Mrs. G. P. Bassett, Jr., Mrs. D. M. Clemson, Mrs. Charles L. Miller, Mrs. V. L. P. Shriver, Mrs. W. A. Bostwick, Mrs. A. C. Dinkey, Mrs. A. P. Childs, Jr., Mrs. F. R. Dravo, Mrs. Herman Griffin, Mrs. W. H. Nimmick, Mrs. F. H. Lloyd, Mrs. S. G. Cooper, Mrs. S. H. Church, Mrs. E. N. Ohl.

The Institute Headquarters was established at the Hotel Schenley, where a bureau of information was maintained for the benefit of the visiting members and guests of the party during the time of the meeting.

TECHNICAL SESSIONS.

The first session, held Tuesday evening, Mar. 1, 1910, in the large lecture hall of the Carnegie Institute, was called to order by Harrison W. Craver, Secretary of the Local Committee. Mr. Craver introduced Dr. John A. Brashear, who extended to the members and guests a hearty welcome to the city of Pittsburg, to which David W. Brunton, President of the Institute, responded.

The Secretary announced, with appropriate comment, the death of Dr. Charles B. Dudley, of Altoona, and also that of William Metcalf, of Pittsburg, concerning whose life and work Col. H. P. Bope, of the same city, delivered a brief but eloquent and appreciative address. (More elaborate biographical notices of both of these distinguished officers of the Institute are published in the present volume.)

The following papers were presented in oral abstract by the authors:

Development of Hindered-Settling Apparatus, by Robert H. Richards, Boston, Mass. (Illustrated by lantern-slides.)

A Few Mining-Districts in the Argentine Republic, South America, by Z. S. Beyl.[1] (Illustrated by lantern-slides.)

The second session of the Institute, held at the same place, Wednesday morning, March 2, was called to order by President Brunton.

The following papers were presented in oral abstract by the authors:

Systematic Exploitation in the Pittsburg Coal-Seam, by F. Z. Schellenberg, Pittsburg, Pa. Discussed by Frank W. De Wolf, Urbana, Ill.[2]

*A Commercial Fuel-Briquette Plant, by William H. Blauvelt, Syracuse, N. Y. Discussed by C. T. Malcolmson, Chicago, Ill.; E. W. Parker, Washington, D. C.; John Birkinbine, of Philadelphia, Pa.; and R. W. Raymond, New York, N. Y.

*Dust-Explosions in Coal-Mines, by George Samuel Rice, Pittsburg, Pa.

The Gaseous Decomposition-Products of Black Powder, by C. M. Young, Lawrence, Kan.

The Combustion of Coal, by Joseph A. Holmes, Washington, D. C., and Henry Kreisinger, Pittsburg, Pa.

The third session of the Institute, held at the same place, Wednesday evening, March 2, was called to order by President Brunton.

The following papers were presented in oral abstract by the authors:

The Conditions of Accumulation of Petroleum in the Earth, by David T. Day, Washington, D. C.

The School of Mines at the Pittsburg University, by M. E. Wadsworth, Pittsburg, Pa.[1]

The Technical Schools of the Carnegie Institution, by Fred Crabtree, Pittsburg, Pa.[1]

Improvement in Cyanide Practice, by E. Gybbon Spilsbury, New York, N. Y.

The Dwight and Lloyd Sintering-Process, by Arthur S. Dwight, New York, N. Y. (Illustrated by lantern-slides.)[1]

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* Distributed in printed form.

[1] Not furnished for publication.

[2] Discussion not furnished for publication.

The fourth and concluding session, held at the same place, Friday morning, March 4, was called to order by President Brunton.

The following papers were presented in oral abstract by the authors:

The Huronian as a Gold-bearing Terrane, by Robert Bell, Ottawa, Canada.[3]

The Action of Explosives in Rocks of Different Degrees of Hardness, by W. O. Snelling, Pittsburg, Pa.[3]

Introduction of the Thomas Basic Steel Process in the United States, by George W. Maynard, New York, N. Y.

Electric Mine-Hoists, by D. B. Rushmore and K. A. Pauly, Schenectady, N. Y. (Illustrated by lantern-slides.)

Field-Investigations of Structural Materials by the U. S. Geological Survey, by E. F. Burchard, Washington, D. C.

Exploration of Certain Iron-Ore and Coal-Deposits in the State of Oaxaca, Mexico, by J. L. W. Birkinbine, Mexico City, Mexico.

The Chemical Control of Slimes, by Harrison E. Ashley, Pittsburg, Pa.

In addition to the papers already noted, the following were read by title for future publication:

Combustion in Cement-Burning, by Byron E. Eldred, New York, N. Y.

*A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest, by John B. Dilworth, Philadelphia, Pa.

*Cyanide-Plant and Practice at the Minas del Tajo, Rosario, Sinaloa, Mexico, by George A. Tweedy and Roger L. Beals, Rosario, Sinaloa, Mexico.

Tests of an Ilgner Electric Hoist, by R. R. Seeber, Winona, Mich.

Bibliography to Accompany Paper on Electric Mine-Hoists, by D. B. Rushmore and K. A. Pauly, Schenectady, N. Y.

The Copper-Provinces of the United States, by William H. Emmons, Chicago, Ill.[3]

*The Behavior of Copper-Slags in the Electric Furnace, by Lewis T. Wright, San Francisco, Cal.

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* Distributed in printed form.

[3] Not furnished for publication.

Chemical Laboratories in Iron- and Steel-Works, by George W. Maynard, New York, N. Y.

A Portable Assay-Outfit for Field-Work, by S. K. Bradford, National, Nev.

*The Girod Electric Furnace, and the French Works Using the Paul Girod Steel-Process, by Wilhelm Borchers, Aachen, Germany.

Improvements in Blast-Roasting, by Herbert Haas, San Francisco, Cal.[4]

Ultimate Source of Ores, by Charles R. Keyes, Des Moines, Iowa.

*The Genesis of the Leadville Ore-Deposits, by Max Boehmer, Denver, Colo.

Heats of Formation of Some Ferro-Calcic Silicates, by H. O. Hofman and C. Y. Wen, Boston, Mass.

*Professional Ethics, by R. W. Raymond, New York, N. Y.

*Application of Descriptive Geometry to Mining-Problems, by Joseph W. Roe, New Haven, Conn.

*The Behavior of Copper-Matte and Copper-Nickel Matte in the Bessemer Converter, by David H. Browne, Copper Cliff, Ontario, Canada.

*Mining-Conditions in the Belgian Congo (Congo Free State), by Sydney H. Ball, New York, N. Y., and Millard K. Shaler, Lawrence, Kan.

*Sampling Anode-Copper, with Special Reference to Silver-Content, by William Wraith, Anaconda, Mont.

The Giroux Shaft at Kimberly, Nev., by C. Everard Arnold, Kimberly, Nev.

*The Combustion-Temperature of Carbon and Its Relation to Blast-Furnace Operation, by Clarence P. Linville, State College, Pa.

*The Fushun Colliery, South Manchuria, by Warden A. Moller, Tientsin, China.

Federal Coal-Mines in the Philippines, by Oscar H. Reinholt, Pasadena, Cal.[4]

*Professional Ethics, by Victor G. Hills, Denver, Colo.

The Form of Coal-Analysis Best Adapted for Comparative Purposes, by M. R. Campbell, Washington, D. C.[4]

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* Distributed in printed form.

[4] Not furnished for publication.

Some Experimental Work on Mining-Education, by M. E. Wadsworth, Pittsburg, Pa.[5]

Discussion of the paper of Edmund D. North, Glass Mine-Models, by A. Scott Reid, London, England.[6]

Discussion of the paper of Audley H. Stow, Pressure-Fans *vs.* Exhaust Fans, by R. V. Norris, Wilkes-Barre, Pa.[7]

*Discussion of the paper of Lewis T. Wright, Metal-Losses in Copper-Slags, by J. Parke Channing, New York, N. Y.

*Discussion of the paper of Charles R. Keyes, Borax-Deposits of the United States, by A. M. Strong, Bishop, Cal.[8]

*Discussion of the paper of E. W. Parker, The Conservation of Coal in the United States, by W. L. Saunders, New York, N. Y.[9]

Discussion of the paper of Charles R. Keyes, Ozark Lead- and Zinc-Deposits. Mr. Keyes's reply to Mr. Buckley.

Discussion of the paper of Charles R. Keyes, Genesis of the Lake Valley, New Mexico, Silver-Deposits. Mr. Keyes's reply to Messrs. Courtis and Macdonald.[10]

Discussion of the paper of James Douglas, Conservation of Natural Resources, by James Douglas, New York, N. Y.[11]

Discussion of the paper of Albert F. J. Bordeaux, The Cyaniding of Silver-Ores in Mexico, by Herbert A. Megraw, San Luis de la Paz, Guanajuato, Mexico.[12]

Discussion of the paper of H. O. Hofman and C. R. Hayward, Pan-Amalgamation: an Instructive Laboratory-Experiment, by George W. Riter, Salt Lake City, Utah, and reply by Messrs. Hofman and Hayward.[13]

During the sessions there was exhibited in the lecture-hall for the first time an exceedingly good portrait in oil by Farley, a Philadelphia artist, of the late Thomas M. Drown, for many years Secretary, and in 1897 President of the Institute. This portrait was presented to the Institute by James Gayley, Dr. James Douglas, and Dr. R. W. Raymond.

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* Distributed in printed form.

[5] Not furnished for publication.

[6] *Trans.*, xl., 913 (1910).

[7] *Idem*, p. 874.

[8] *Idem*, p. 909.

[9] *Idem*, p. 901.

[10] *Idem.*, p. 833.

[11] *Idem*, p. 878.

[12] *Idem*, p. 917.

[13] *Idem*, p. 865.

Excursions and Entertainments.

An account of the excursions and entertainments in connection with the Pittsburg meeting was published in *Bulletin* No. 40, April, 1910, pp. 315 to 334.

Members and Guests in Attendance at the Sessions and Excursions.

Acker, Louis K., Jr., Pittsburg, Pa.
Allderdice, Taylor, Pittsburg, Pa.
Allderdice, Mrs. Taylor, Pittsburg, Pa.
Ashley, Harrison E., Pittsburg, Pa.
Austin, W. W., St. Louis, Mo.
Bassett, Mrs. G. P., Pittsburg, Pa.
Beik, Emile, New York, N Y.
Bell, Robert, Ottawa, Can.
Bell, Miss, Ottawa, Can.
Bellinger, H. P., Syracuse, N. Y.
Bellinger, Mrs. H. P., Syracuse, N. Y.
Beyl, Z. S., Buenos Aires, Argentine Republic, S. A.
Bibbins, J. R., Pittsburg, Pa.
Birkinbine, J. L. W., Philadelphia, Pa.
Birkinbine, John, Philadelphia, Pa.
Blauvelt, W. H., Syracuse, N. Y.
Bleininger, Dr. A. V., Pittsburg, Pa.
Bope, Col. H. P., Pittsburg, Pa.
Bostwick, W. A., Pittsburg, Pa.
Bostwick Mrs. W. A., Pittsburg, Pa.
Brashear, Dr. John A., Pittsburg, Pa.
Brown, G. H., Pittsburg, Pa.
Brunton, D. W., Denver, Colo.
Buhl, William, Jr., Pittsburg, Pa.
Burchard, E. F., Washington, D. C.
Callbreath, J. F., Jr , Denver, Colo.
Cavanaugh, J. R., Pittsburg, Pa.
Childs, Mrs. A. P., Jr., Pittsburg, Pa.
Clark, H. H., Pittsburg, Pa.
Clemson, Mrs. D. M , Pittsburg, Pa.
Cole, W. R., Philadelphia, Pa.
Collord, George L., Sharpsville, Pa.
Conrad, H. V., Pittsburg, Pa.
Conrad, Mrs. H. V., Pittsburg, Pa.
Corey, A. A., Pittsburg, Pa.
Crabtree, Fred, Pittsburg, Pa.
Craver, Harrison W., Pittsburg, Pa.
Crawford, R. C., Pittsburg, Pa.
Crawford, Mrs. R. C., Pittsburg, Pa.
Cooper, Mrs. S. G., Pittsburg, Pa.
Custer, L. R., Pittsburg, Pa.
Davidson, G. M., Chicago, Ill.
Day, David T., Washington, D. C.
De Wolf, F. W., Urbana, Ill.
Dinkey, A. C., Pittsburg, Pa.
Dinkey, Mrs. A. C., Pittsburg, Pa.
Dravo, Mrs. F. R., Pittsburg, Pa.
Dwight, A. S., New York, N. Y.
Dwight, Mrs. A. S., New York, N. Y.
Dwight, Theodore, New York, N. Y.
Elmer, Wm., Pittsburg, Pa.
Emley, W. E., Pittsburg, Pa.
Fairbairn, C. T., Pittsburg, Pa.
Gates, Elvin R., Pittsburg, Pa.
Gluck, George, Pittsburg, Pa.
Godley, George McM., New York, N.Y.
Goodale, S. L., Pittsburg, Pa.
Griffin, Mrs. Herman, Pittsburg, Pa.
Grose, J. H., Pittsburg, Pa.
Haldeman, George T., Pittsburg, Pa.
Hall, Clarence, Pittsburg, Pa.
Hansell, N. V., New York, N. Y.
Hart, R. B., Pittsburg, Pa.
Hartzell, L. M., Pittsburg, Pa.
Hazlewood, A. J., Pittsburg, Pa.
Henderson, W. M., Pittsburg, Pa.
Herr, Mrs. B. M., Pittsburg, Pa.
Hibbard, Henry D., Plainfield, N. J.
Holmes, J. A., Washington, D. C.
Horton, James, Pittsburg, Pa.
Howell, Spencer F., Pittsburg, Pa.
Huntoon, L. D., New Haven, Conn.
Hursh, R. K., Pittsburg, Pa.
Jones, J. H., Pittsburg, Pa.
Jones, W. L., Pittsburg, Pa.
Kann, W. L., Pittsburg, Pa.
Kaplan, Harry, Pittsburg, Pa.
Kennedy, Julian, Pittsburg, Pa.
Kennedy, Mrs. Julian, Pittsburg, Pa.
Kirchhoff, C., New York, N. Y.
Lawler, J. C., Pittsburg, Pa.
Le Boutillier, Clem., High Bridge, N. J.
Lee, J. Henry, Baltimore, Md.
Lee, Mrs. J. Henry, Baltimore, Md.
Le Fevre, S., Mineville, N. Y.
Lincoln, L. P., Pittsburg, Pa.
Linville, C. P., State College, Pa.

Lloyd, Mrs. F. H., Pittsburg, Pa.
Lyon, D. A., Heroult, Cal.
McCormick, R. T., Pittsburg, Pa.
McNaugher, D. W., Pittsburg, Pa.
MacClure, C. A., Baltimore, Md.
MacClure, Mrs. C. A., Baltimore, Md.
Malcolmson, C. T., Chicago, Ill.
Marsh, Mrs. J. W., Pittsburg, Pa.
Maynard, George W., New York, N. Y.
Means, E. C., Low Moor, Va.
Means, Mrs. E. C., Low Moor, Va.
Meller, H. B., Pittsburg, Pa.
Mellon, Mrs. T. A., Pittsburg, Pa.
Messler, E. L., Pittsburg, Pa.
Mesta, George, Pittsburg, Pa.
Miller, Charles L., Pittsburg, Pa.
Miller, Mrs. Charles L., Pittsburg, Pa.
Nimick, Mrs. W. H., Pittsburg, Pa.
Ohl, Mrs. E N., Pittsburg, Pa.
Orchard, Charles, Pittsburg, Pa.
Ormrod, George, Allentown, Pa.
Ormrod, John D., Allentown, Pa.
Orton, Bror, Stockholm, Sweden.
Packer, W. H., Pittsburg, Pa.
Pargny, E. W., Pittsburg, Pa.
Parker, E. W., Pittsburg, Pa.
Pauly, Karl A., Schenectady, N. Y.
Phillips, Francis C., Pittsburg, Pa.
Porterfield, H. A., Pittsburg, Pa.
Raymond, R. W., Brooklyn, N. Y.
Rea, W. H., Pittsburg, Pa.
Rice, George S., Pittsburg, Pa.
Richards, Robert H., Boston, Mass.
Riley, G. H., Pittsburg, Pa.
Rogers, John V., Jr., Philadelphia, Pa.
Rösler, Dr., New York, N. Y.
Ross, H Earle, Pittsburg, Pa.
Saunders, W. L., New York, N. Y.
Scaife, W. L., Pittsburg, Pa.
Schellenberg, F. Z., Pittsburg, Pa.
Sherrerd, John M., High Bridge, N. J.
Shriver, Mrs. V. L. P., Pittsburg, Pa.
Smith, Franklin W., Bisbee, Ariz.
Smith, Lloyd B., Pittsburg, Pa.
Smith, Oberlin, Bridgeton, N. J.
Smith, Mrs. Oberlin, Bridgeton, N. J.
Snelling, Walter O., Washington, D. C.
Speller, F N, Pittsburg, Pa.
Spilsbury, E. G., New York, N. Y.
Spilsbury, Miss B. G., New York, N. Y.
Spilsbury, Miss R. F., New York, N. Y.
Stanton, F. McM., New York, N. Y.
Stoek, H. H., Urbana, Ill.
Street, G. B., Wilmington, Del.
Stoughton, Bradley, New York, N. Y.
Struthers, Joseph, New York, N. Y.
Taylor, Knox, High Bridge, N. J.
Taylor, S. A., Pittsburg, Pa.
Tolstead, E. B., Pittsburg, Pa.
Vogel, Felix A., New York, N. Y.
Wright, Charles L., Pittsburg, Pa.
Wadsworth, M. E., Pittsburg, Pa.
Watson, R. H., Pittsburg, Pa.
Wales, S. S., Pittsburg, Pa.
Weil, T., Pittsburg, Pa.
Wellman, S. T., Cleveland, Ohio.
Wellman, Mrs. S. T., Cleveland, Ohio.
Wilson, E. R., Scranton, Pa.
Young, C. M., Lawrence, Kan.

Proceedings of the Ninety-Ninth Meeting, Canal Zone, November, 1910.

COMMITTEES.

CANAL ZONE.—Col. George W. Goethals, U. S. A., *Chairman;* Lt.-Col. H. F. Hodges, U. S. A.; Lt.-Col. D. D. Gaillard, U. S. A.; Lt.-Col. William L. Sibert, U. S. A.; H. H. Rousseau, U. S. N.; Col. W. C. Gorgas, U. S. A.; Maurice H. Thatcher; Joseph Bucklin Bishop; Lt.-Col. Charles F. Mason, U. S. A.; S. B. Williamson; Major Eugene T. Wilson, U. S. A.; Edward Schildhauer; Henry Goldmark; Edward C. Sherman; W. G. Comber; W. B. Corse; J. A. Walker; Joseph A. Le Prince; J. A. Smith; Lt. Frederick Mears, U. S. A.

HAVANA.—Y. Y. Polledo, *Chairman;* Diego Lombillo Clark, José Primelles, J. R. Villalon, José Artola, J. Ducassi, Luis Morales, Enrique Montonlieu.

ENTERTAINMENT ON SHIPBOARD.—William L. Saunders, *Chairman;* Mr. and Mrs. S. D. Warriner, Mr. and Mrs. Thomas Robins, Mr. and Mrs. George D. Barron, Mr. and Mrs. R. B. Watson, W. E. Greene.

GENERAL MANAGEMENT.—Joseph Struthers, *Chairman;* Albert E. Vaughan.

There were ten sessions of the Institute held for the presentation and discussion of technical or professional papers. Eight of these sessions were held on the steamer while *en route*, and two in the ball-room of the Hotel Tivoli, at Ancon, C. Z. Mr. David W. Brunton, President of the Council, presided at all the sessions, except the sixth, at which Mr. Charles Kirchhoff, past President, occupied the chair.

The following list includes the papers presented in oral abstract by the authors:

First Session, October 24.

The Summit Hill Mine-Fire, by W. A. Lathrop, Philadelphia, Pa.

Mine-Fire at the Calumet and Hecla Mine, No. 8 Shaft, by S. D. Warriner, Wilkes-Barre, Pa.

Fire in the Big Lick Stope, Lykens Valley and Williamstown Colliery, by R. V. Norris, Wilkes-Barre, Pa.

Mine-Fire in the Heckscherville Valley, Schuylkill Field, Pa., by W. J. Richards, Pottsville, Pa.

Fire in the Monarch Mine, near Rock Springs, Wyo., by S. A. Taylor, Pittsburg, Pa.

Fire in the Anaconda Mine, Butte, Mont., by David W. Brunton, Denver, Colo.

Fire in the Leonard Mine, Butte, Mont., by Charles W. Goodale, Butte, Mont.

Second Session, October 27.

Mine-Fire in the Luke Fidler Colliery, Shamokin, Pa., by R. V. Norris, Wilkes-Barre, Pa.

Four Fires at the Vulcan Mine, Vulcan, Mich., by William Kelly, Vulcan, Mich.

Results of the Recent Explosion in the Esperanza Mine, Coahuila, Mexico, by Edward W. Parker, Washington, D. C.

Fire in the Kimberley Mine, South Africa, by Gardner F. Williams, Washington, D. C.

Absence of Fires in the Gold-Mines of the Rand, by Hennen Jennings, Washington, D. C.

Third Session, October 28.

Production of Pig-Iron by the Electric Furnace in Sweden, by Joseph W. Richards, South Bethlehem, Pa.

Fourth Session, October 31.

History and Construction of the Panama Canal, by W. L. Saunders, New York, N. Y.

Panama and Its People, by John M. Sherrerd, Easton, Pa.

Fifth Session, November 4. *Hotel Tivoli, Ancon.*

The Work of the Sanitation Department on the Isthmus, by Col. W. C. Gorgas, member of the Isthmian Canal Commission.

The Geology of the Isthmus (an informal talk), by Willard P. Hayes, Washington, D. C.

The Diamond-Mines of South Africa, and the Gold-Mines of the Rand (Illustrated), by Gardner F. Williams, Washington, D. C.

[SECRETARY'S NOTE.—The contributions of the first five sessions were made by the authors on the spur of the moment, and with the implied understanding that they would not necessarily be called for in written form for publication. It is hoped, however, that many of these interesting impromptu statements may be secured for the *Transactions*, so that the whole membership of the Institute may share, at least partly, in the thrilling interest and professional instruction which they afforded to their hearers on the *Prinz August Wilhelm*.—R. W. R.]

Sixth Session, November 6. *Hotel Tivoli, Ancon.*

*Recent Developments in the Undercutting of Coal by Machinery (Illustrated), by Edward W. Parker, Washington, D. C.

Manufacture of Electrical Mining-Machinery (Illustrated), by David B. Rushmore, Schenectady, N. Y.[1]

An informal talk, with lantern-illustrations, on the scenery and big game of South Africa, by Gardner F. Williams, Washington, D. C.[1]

Seventh Session, November 9.

The Panama Canal. General discussion. (This discussion, including contributions subsequently received, was printed in *Bulletin* No. 49, January, 1911, pp. 35 to 118.)

Eighth Session, November 11.

Recent Water-Power Developments in Montana, and the Uses of Electric Power in Pumping, Compressing, and Hoisting, by Charles W. Goodale, Butte, Mont.[1]

Ninth Session, November 12.

Conservation in the Preparation of Anthracite for the Market, by W. S. Ayres, Hazleton, Pa.[1]

Report of the Delegates of the Institute Attending the American Mining Congress at Los Angeles, by D. W. Brunton, Denver, Colo.[2]

*The Report of the Committee on Uniform Mining-Laws for the Prevention of Mine-Accidents, by W. R. Ingalls, J. Parke Channing, James Douglas, and John Hays Hammond, New York, N. Y., and James R. Finlay, Goldfield, Nev.[3] Followed by a general discussion.[1]

Tenth Session, November 14.

The Panama Canal. Continued discussion, resulting in the adoption of the statement printed below, which, while it cannot be regarded, under our constitution, as an official utterance

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* Distributed in printed form.

[1] Not furnished for publication.

[2] *Bulletin* No. 48, December, 1910, pp. v to xi.

[3] *Idem*, No. 46, October, 1910, pp. 839 to 888.

of the Institute, expresses the unanimous opinion of those members who signed it.

S. S. PRINZ AUGUST WILHELM, AT SEA,
November 14, 1910.

We, the undersigned, members and guests of the American Institute of Mining Engineers, after a visit to the Isthmus of Panama, and inspection of the work of the United States Isthmian Canal Commission, and after full discussion of our individual impressions, find ourselves in unanimous agreement as to the following conclusions:

1. The present plan of the work is clearly practicable, and the best in our judgment that could be devised under the conditions imposed. It is perhaps a question whether by the choice of a higher level some of the difficulties and uncertainties of excavation in the Culebra Cut might not have been minimized; but a higher level has its disadvantages also; and no one now seriously proposes such a plan. On the other hand, we are convinced that a canal at a lower level, and especially at sea-level, is practically out of the question; that no man can estimate its cost, or even guarantee its satisfactory completion and maintenance at any cost. We are satisfied that the sea-level canal, as proposed, if actually completed, would be inferior to the present lock canal, by reason of its necessarily narrow and tortuous channel, its liability to many disturbances from which the lock canal is comparatively free, etc. The experience gained in the Culebra Cut throws additional light upon the sea-level plan, and renders that scheme less worthy of approval by engineers than it was when with less information some eminent authorities favored it. In a word, we do not think that any prudent engineer would now recommend the deepening of the Culebra Cut below the level now fixed for it.

The creation of the great Gatun lake by means of the Gatun dam seems to us to be the best possible way of dealing with the floods of the Chagres and other streams. The location of the Gatun dam, spillway, and locks is singularly favorable for such constructions; and there is, in our judgment, no reason for any anxiety as to their stability.

The one serious remaining problem is presented by the nature of the ground in the Culebra Cut. There have been extensive slides on the sides of this excavation, and more of them are to be expected; but they involve nothing more than the cost and delay of removing the material which they will force into the cut. They will ultimately end, and we regard as reasonable the calculation of the engineers in charge as to the time and money which they may call for. The results of these calculations are included in the estimates of the Commission as to the cost of the canal and the date of its completion.

2. We are unanimous in our praise of the manner in which sanitation, excavation, transportation, and construction are performed with rapidity, skill, and economy. A spirit of loyalty, emulation, industry, and pride seems to animate employees and officers alike. This spirit, so difficult to arouse among workers in tropical climates, is due in this case to two causes: first, the inspiring example of Colonel Goethals and his associates, and secondly, the splendid work of the sanitation department under Colonel Gorgas. The cities of Panama and Colon, though politically outside of the Canal Zone, have shared in the benefits of the sanitary administration, and reflect an unwonted cleanliness, comfort, and safety.

3. We acknowledge the entire freedom and fullness with which everything we

desired to see was shown to us, and everything we desired to know was told us, by the officers of the Commission. There was evidently no wish to withhold or conceal anything. On the contrary, inquiry and criticism were frankly sought and heartily welcomed.

This is but a meager summary of the points on which we are agreed. The details of individual opinion will appear later in the published report of our discussions. Meanwhile, we unite in this common declaration, which covers our conclusions on all main points. We think the present plan of the canal is good; that the work is in thoroughly capable hands; that it is progressing satisfactorily, and that it will be completed by the date set for it, January 1, 1915, and probably earlier, provided Colonel Goethals and his associates receive the hearty support of the American people, and its representatives in Congress. The canal engineers are the right men in the right place. The great work in which they are engaged is not connected with partisan politics, and citizens of all parties should combine to secure its early and triumphant completion. In that consummation every American should take greater pride than in any victory of military or political conflict.

(SIGNATURES.)

D. W. Brunton, W. L. Saunders, R. W. Raymond, Joseph Struthers, William Kelly, R. V. Norris, Joseph W. Richards, Henry S. Drinker, W. E. C. Eustis, C. W. Goodale, William Kent, Edward W. Parker, Walter Wood, W. S. Ayres, George D. Barron, Thomas E. Brown, A. C. Carson, Josiah H. Clark, F. L. Clerc, Torbert Coryell, James S. Cunningham, Will Ward Duffield, Howard N. Eavenson, Augustus H. Eustis, H. W. Hardinge, Rowland F. Hill, Hennen Jennings, J. Elmer Jones, W. A. Lathrop, A. F. Lucas, Eugene McAuliffe, J. Gibson McIlvain, Walter T. Page, W. J. Richards, D. M. Riordan, Thomas Robins, David B Rushmore, F. W. Scarborough, Samuel A. Taylor, George H. Warren, S. D. Warriner, R. B. Watson, H. A. J. Wilkens, Gardner F. Williams, Howard Wood, Thomas D. Wood, John W. Ailes, William I. Berryman, Alexander L. Brodhead, W. J. Davidson, D. C. Dodge, John W. Donnan, Philip Goodwill, William Ellery Greene, C. B. Houck, Bedford Leighton, W. F. Mackay, D. G. Miller, Frank P. Miller, T. H. Miller, D. G Moore, Thomas W. Orbison, C. M. Russell, Robert C. Sahlin, Frederick R. Sayen, F. L. Schoew, S. H. Sherrerd, W. S. Stewart, Charles S. Thomas, Michael Tracy, Joseph Underwood, A. E. Vaughan, Frank M. Warren, Edwin L. Watson, H. M. Weaver, Hugo Weinberger, William Wilkie.

The Manufacture and Refining of Steel in the Electrical Furnace, by Joseph W. Richards, South Bethlehem, Pa.[4]

In addition to those above mentioned, the following papers were presented and read by title for future publication:

†Manganese-Ore in Unusual Form, by William P. Blake.

*Crushing-Machines for Cyanide-Plants, by Mark R. Lamb, Milwaukee, Wis.

* Distributed in printed form.

† In proof, available for consultation.

[4] Not furnished for publication.

*Recent Progress in Blast-Roasting, by H. O. Hofman, Boston, Mass.

*The Nicola Valley Coal-Field, British Columbia (Postscript), by Milnor Roberts, Seattle, Wash.[5]

*Labor-Saving Appliances in the Assay-Laboratory, by Edward Keller, Perth Amboy, N. J.

‡Dry-Washing for Placer-Gold in Sonora, Mexico, by J. V. Richards, Spokane, Wash.

*Method of Determining the Meridian from a Circumpolar Star at Any Hour, by E. R. Rice, Wickenburg, Ariz.

†Pyritic Smelting in Leadville, by Charles H. Doolittle, Salt Lake City, Utah, and Royal P. Jarvis, Knoxville, Tenn.

†The Laws of Intrusion, by Blamey Stevens, Valdez, Alaska.

*Biographical Notice of Franklin R. Carpenter, by H. O. Hofman, Boston, Mass.

*The Reduction of Calcium Sulphate by Carbon Monoxide and Carbon, and the Oxidation of Calcium Sulphide, by H. O. Hofman and W. Mostowitsch, Boston, Mass.

*Biographical Notice of William Phipps Blake, by R. W. Raymond, New York, N. Y.

†Tops of Copper Blast-Furnaces, by N. H. Emmons, Copperhill, Tenn.

Mining in Nicaragua, by T. Lane Carter, Chicago, Ill.

‡Mine-Survey Notes, by George W. Riter, Salt Lake City, Utah.

‡The Solid Non-Metallic Impurities in Steel (Sonims), by Henry D. Hibbard, Plainfield, N. J.

‡Theory of Dust-Explosions, by Audley H. Stow, Maybeury, W. Va.[6]

*The Gold-Fields of French Guiana, and the New Method of Dredging, by Albert F. J. Bordeaux, Thonon-Les-Bains, Savoie, France.

*The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold-Deposits in the United States, by William H. Emmons, Chicago, Ill.[7]

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* Distributed in printed form.
† In proof, available for consultation.
‡ In manuscript, available for consultation.

[5] *Trans.*, xl., 803 (1910).

[6] Held for vol. xlii.

[7] *Bulletin* No. 46, October, 1910, pp. 767 to 837. Held for vol. xlii.

The Conference Department at Lehigh University, by Dr. Henry S. Drinker, South Bethlehem, Pa.

The Condensation of Fume and The Neutralization of Furnace-Gases, by F. T. Havard, Madison, Wis.

*Discussion of paper of George W. Maynard, Introduction of the Thomas Basic Steel Process in the United States, by Henry D. Hibbard, Plainfield, N. J.

†Discussion of paper of Max Boehmer, The Genesis of the Leadville Ore-Deposits, by W. Morton Webb, Germiston, Transvaal, South Africa.

‡Discussion of paper of Joseph A. Holmes, The Combustion of Coal, by Prof. William Kent, Montclair, N. J.

‡Discussion of paper of John B. Dilworth, A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest, by Frank Firmstone, Easton, Pa.

*Discussion of paper of W. H. Blauvelt, A Commercial Fuel-Briquette Plant, by Robert Schorr, San Francisco, Cal.

*Discussion of paper of Byron E. Eldred, Combustion in Cement-Burning, by Robert Schorr, San Francisco, Cal., and Byron E. Eldred, Tuckahoe, N. Y.

‡Discussion of paper of Mark R. Lamb, Crushing-Machines for Cyanide-Plants, by Herbert A. Megraw, San Luis de la Paz, Guanajuato, Mexico.

‡Discussion of paper of H. O. Hofman, Recent Progress in Blast-Roasting by James W. Neill, Pasadena, Cal.

Excursions and Entertainments.

An account of the excursions and entertainments in which the members and guests of the Institute participated at Havana, Cuba; Kingston, Jamaica; the Canal Zone, and Port Limon and San José, Costa Rica, was published in *Bulletin* No. 48, December, 1910, pp. 1022 to 1054.

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* Distributed in printed form.

† In proof, available for consultation.

‡ In manuscript, available for consultation.

List of Members and Guests Constituting the Excursion Party.

Ailes, J. W., Pittsburg, Pa.
Ayres, Mrs. E. L. C., Bound Brook, N. J.
Ayres, W. S., Hazleton, Pa.
Ayres, Mrs. W. S., Hazleton, Pa.
§ Barron, George D., Rye, N. Y.
Barron, Mrs. George D., Rye, N. Y.
Barron, Miss Dorothy, Rye, N. Y.
Barron, Miss M. Elena, Rye, N. Y.
§ Berryman, W. I., Pittsburg, Pa.
Brodhead, Alexander, Catasauqua, Pa.
Brown, Thomas E., New York, N. Y.
Brunton, David W., Denver, Colo.
Carson, Arthur C., New York, N. Y.
Clark, Josiah H., New York, N. Y.
Clerc, F. L., Boulder, Colo.
Coryell, Torbert, Lambertville, N. J.
Coryell, Mrs. T., Lambertville, N. J.
Cunningham, J. S., Johnstown, Pa.
Cunningham, Mrs. J. S., Johnstown, Pa.
Davidson, W. I., Port Richmond, S.I., N.Y.
Dodge, Col. D. C., Denver, Colo.
Dodge, Mrs. D. C., Denver, Colo.
Donnan, J. W., Port Richmond, S.I, N.Y.
Drinker, Dr. H. S., S. Bethlehem, Pa.
Drinker, Mrs. H. S., S. Bethlehem, Pa.
§ Duffield, W. W., Harlan, Ky.
Eavenson, Howard N., Gary, W. Va.
Eavenson, Mrs. H. N., Gary, W. Va.
Eavenson, Miss S. I., Gary, W. Va.
Eustis, Augustus H, Boston, Mass.
Eustis, W. E. C., Boston, Mass.
Goodale, Charles W., Butte, Mont.
Goodwill, Philip, Bramwell, W. Va.
Goodwill, Mrs. Philip, Bramwell, W. Va.
Greene, William E., Cleveland, Ohio.
Hardinge, H. W, New York, N. Y.
Hardinge, Mrs. H. W., New York, N. Y.
Hardinge, Miss, New York, N. Y.
§ Hill, Rowland F., Pulaski, Va.
Houck, C. B., Hazleton, Pa.
‡ James, Harry C., Denver, Colo.
Jennings, Hennen, Washington, D C.
Jones, J. Elmer, Hazleton, Pa.
Kelly, William, Vulcan, Mich.
Kelly, Mrs. William, Vulcan, Mich.
Kent, Prof. William, Montclair, N. J.
Kirchhoff, Charles, New York, N. Y.
Lathrop, William A., Philadelphia, Pa.
Lathrop, Mrs. W. A., Philadelphia, Pa.
Leighton, Bedford, Binghamton, N. Y.
* Lucas, A. F., Washington, D. C.
McAuliffe, Eugene, Chicago, Ill.
McAuliffe, Mrs Eugene, Chicago, Ill.
McIlvain, J. G., Philadelphia, Pa.
McIlvain, J. G., Jr., Philadelphia, Pa.
Mackay, H. F., Cleveland, Ohio.
Miller, David G., Denver, Colo.
Miller, F. P., Philadelphia, Pa.
Miller, Theo. H., Poughkeepsie, N. Y.
Moore, D. G., Elizabeth, N. J.
Norris, Robert V., Wilkes Barre, Pa.
Orbison, Thomas W., Appleton, Wis.
Orbison, Mrs. T. W., Appleton, Wis.
Page, Walter, Omaha, Neb.
Page, Mrs. Walter, Omaha, Neb.
Parker, Edward W., Washington, D. C.
Parker, Mrs. E. W., Washington, D. C.
‡ Perkins, Henry C., New York, N. Y.
† Porter, Robert P., London, England.
Raymond, Dr. R. W., New York, N. Y.
Richards, Prof. J. W., S. Bethlehem, Pa.
Richards, W. J., Pottsville, Pa.
Richards, Mrs. W. J., Pottsville, Pa.
Richards, Miss, Pottsville, Pa.
† Riordan, D. M., New York, N. Y.
Robins, Thomas, New York, N. Y.
Robins, Mrs. Thos., New York, N. Y.
Rushmore, D. B., Schenectady, N. Y.
Russell, C. M., Massillon, Ohio.
Russell, Mrs. C. M., Massillon, Ohio.
Sahlin, Robert C., S. Bethlehem, Pa.
Saunders, William L., New York, N. Y.
Sayen, F. R., Philadelphia, Pa.
Scarborough, F. W., Richmond, Va.
Scarborough, Miss, Richmond, Va.
Schoew, F. L., Bramwell, W. Va.
Schoew, Mrs. F. L., Bramwell, W. Va.
Sherrerd, John M., Easton, Pa.
Sherrerd, Mrs. John M., Easton, Pa.
* Sherrerd, Samuel H., Easton, Pa.

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* Joined at Havana.
† Joined at Colon.
‡ Left at Colon.
§ Visited Port Limon and San José, Costa Rica.

Stewart, Dr. Walter, Wilkes-Barre, Pa.
Struthers, Dr. Joseph, New York, N. Y.
Taylor, Samuel A., Pittsburg, Pa.
Thomas, S. C., Jr., Denver, Colo.
* Todd, Miss Jeanne, Conshohocken, Pa.
Tracy, M., New York, N. Y.
Underwood, Joseph, Pittsburg, Pa.
§ Underwood, Thomas, Pittsburg, Pa.
Vaughan, A. E., New York, N. Y.
Warren, F. M., Minneapolis, Minn.
Warren, Mrs F. M., Minneapolis, Minn.
Warren, George H., Minneapolis, Minn.
Warren, Mrs. G. H., Minneapolis, Minn.
Warriner, Samuel D., Wilkes-Barre, Pa.
Warriner, Mrs. S. D., Wilkes-Barre, Pa.
§ Watson, E. L., Worcester, Mass.
§ Watson, Mrs. E. L., Worcester, Mass.
Watson, R. B., New York, N. Y.
Watson, Mrs. R. B., New York, N. Y.
Weaver, H. M., Mansfield, Ohio.
§ Weaver, Mrs. H. M, Mansfield, Ohio.
Weinberger, Hugo, Newcastle, Del.
Wilkens, H. A. J., New York, N. Y.
§ Wilkie, William, Buffalo, N. Y.
Williams, David, New York, N. Y.
Williams, Mrs. David, New York, N. Y.
Williams, Gardner F., Washington, D.C.
* Wood, Howard, Conshohocken, Pa.
* Wood, Miss, Conshohocken, Pa.
Wood, Thomas D., Bryn Mawr, Pa.
Wood, Mrs. Thomas D, Bryn Mawr, Pa.
§ Wood, Walter, Philadelphia, Pa.

* Joined at Havana.
§ Visited Port Limon and San José, Costa Rica.

PAPERS.

Ventilating-System at the Comstock Mines, Nevada.

BY GEORGE J. YOUNG,* RENO, NEV.

(Spokane Meeting, September, 1909.)

CONTENTS.

I. INTRODUCTION.

DR. JOHN A. CHURCH, in his treatise on the Comstock Lode,[1] gave a full and clear account of the conditions of the mine

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* Professor of Mining and Metallurgy, University of Nevada.

[1] *The Comstock Lode: Its Formation and History* (1879).

during the period of greatest activity. The difficulties in the way of deep mining at that time were excessive water and high temperature. The drainage of the mines taxed the financial resources of the mining companies, and the high temperatures restricted the capacity of the miners for underground labor. In spite of these difficulties exploratory work was continued until the financial burden of removing the water became too great. The companies then restricted the work to the upper levels. The years 1883 to 1886 marked the period of cessation of deep mining.

For 12 years thereafter mining was confined to the levels above the Sutro tunnel; and, while the question of drainage was more or less agitated, nothing of importance was done until the formation of the Comstock Pumping Association in 1898. Since that time successive levels have been drained until, at present, on the north-end mines, a depth of 700 ft. below the Sutro-tunnel level has been reached. Mining is now carried on in the Ophir mine from the 1,700- to the 2,200-ft. levels, and a winze is being sunk to make connection with the 2,300-ft. level (the 2,450 ft. of the C. & C. shaft).

In 1903 the Ward Shaft Association was formed to undertake the draining of the Central and Gold Hill group of mines, and at the present time a depth of 2,500 ft. has been reached in the Ward shaft. The situation at the Ward looks promising, and the 3,000-ft. level may be reached in the near future. It is proposed to establish a large pumping-station on this level, and then begin the opening of known ore-bodies and the prospecting for new ones.

The main factors which have contributed to the successful solution of the drainage problem are the Sutro tunnel, the use of the hydraulic elevator as a sinking-pump, the concentration of pumping-units, and cheap electrical power.

The question of high temperature and its control still remains as a minor but important problem. The temperatures encountered in the new workings are as great as those in the earlier periods. The Comstock was considered the hottest group of mines in the world, and I am not aware of similar conditions in any other mining-district. A water-temperature as high as 160° F. has been reported from the Ward shaft, and temperatures below the Sutro tunnel are very high in many instances.

The control of excessive underground temperatures is effected by proper ventilation, or by the provision and distribution of air-currents of sufficient volume. Upon the Comstock of late years considerable attention has been paid to ventilation, and underground conditions have been much improved.

The ventilation of metalliferous mines has received but little attention, and comparatively few data are available. The literature of coal-mining, on the other hand, is replete with data of mine-ventilation, the topic being one of considerable interest even at the present time. In coal-mines, ventilation has for its principal object the removal of explosive gases. To accomplish this, fans are generally employed to force into or exhaust from a mine large volumes of air. In metal-mining, while occurrences of gases are not infrequent, the volume of gas encountered is usually so small as to be unimportant, and it is generally non-explosive. In these cases the purpose of the ventilating-current is to supply pure air to the miners in the different working-places, and to remove the gaseous products of blasting. In but few instances has the ventilating-current to perform the additional function of cooling the mine-workings. In most metal-mines ventilating-currents are established by the use of two shafts, one an upcast, the other a down-cast. The natural elevation of the ground-temperature, due to increased depth, is relied upon to warm the air sufficiently to start an upward current. As depth is attained, the average temperature of the ascending current is raised and the ventilating-efficiency of an upcast shaft increased. Fans and air-pipes are used for the ventilation of pits, shafts, and dead-ends, but are seldom used for the general ventilation of a large mine. The Comstock mines furnish an excellent example of the use of natural ventilation for providing a current of sufficient volume partly to overcome abnormal conditions. That natural ventilation is insufficient and not controllable enough, is evidenced by the fact that at the present time one large fan is in operation and another is being erected. Ventilation by small fans is also very largely used to supplement the effect of the shafts. The purpose of this paper is to give a detailed account of the system of ventilation at present in use at the Comstock mines.

II. Temperatures.

In any system of natural ventilation, surface and underground temperatures play an important part.

1. *Surface-Temperatures and Conditions.*—Virginia City, Nev., does not possess a meteorological station, and consequently no records of the weather-conditions are available. However, the records of the government station at Reno, 18 miles NW., supply this deficiency. Virginia City has an altitude of between 6,000 and 6,500 ft., and Reno 4,553 ft., above sea-level.

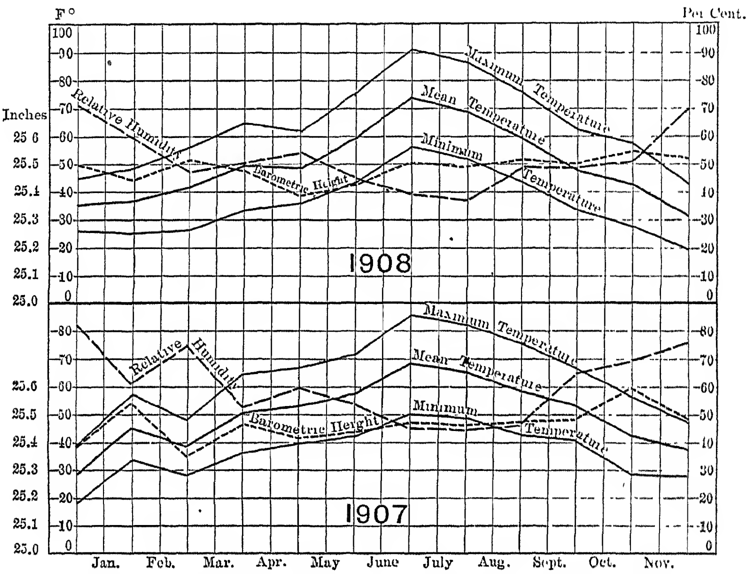

Fig. 1.—Curves of Maximum, Mean, and Minimum Temperatures, Relative Humidity, and Barometic Height, at Reno, Nev., in 1907 and 1908.

The difference in altitude would make some, but not a very great, difference in climatic conditions.

The maximum, mean, and minimum temperatures, the barometric height, and the average relative humidities for the years 1907 and 1908, are plotted in Fig. 1. It should be noted that the day-humidity is much less than the average 24-hr. humidity; for instance, in July, 1907, the average humidity of four days was 23; in August, the average of four days was 19; in September, the average of two days was 21.5; in October, the average of four days was 51; in November, the average of three days was 43; and in December, the average of four days

was 64 per cent. The average day-humidity during the months of July, August, and September was 21; and for the months of October, November, and December it was 51 per cent. The air during the summer and fall months is characterized by a low day-humidity, and during the winter months the day-humidity is about normal. The average yearly temperature is about 50° F.

2. *Underground Temperatures.*—Becker[2] has given excellent data on the rock-temperatures prevailing at different points underground, and to this authority the reader is referred. Air-temperature only will be considered in this paper.

Records of air-temperature in the Ophir mine have been kept for some time, and from these records have been compiled Tables I., II., III., and IV.

TABLE I.—*Temperatures at Sutro-Tunnel Level.*

Date, 1908.	No 1, South of Cross-cut to C & C. Shaft.	No. 2, South of Winze.	No. 3, Head of Winze.	No. 4, From Union Connection to Ophir Incline.
	Degrees F.	Degrees F.	Degrees F.	Degrees F.
June 4.......	85	102	104	100
June 5	90	102	102	100
June 6.......	87	100	104	98
June 8.......	90	100	100	90
June 10.......	92	100	104	.....
June 11.......	94	100	.....	.....
June 15.......	94	98	.....	98
June 24.......	96	.....	.....	.....
July 2.......	96	102	102	96
July 8.......	96	100	.....	.....
July 16.......	97	102	104	97
July 17.......	98	102	105	96

Table I. shows the temperatures measured in the north lateral of the Sutro tunnel during June and July, 1908. The hottest months are July and August, during which the conditions in the main Sutro tunnel in the vicinity of the Combination shaft are particularly trying. No. 1 station in the north lateral (about 500 ft. south of the connection to the C. & C. shaft) shows an average temperature of 91° F. for June and 96.7° for July. Only a feeble air-current was moving towards the Sutro tunnel at this station. At No. 2 station, between

[2] Geology of the Comstock Lode and the Washoe District, *Monograph III.*, *U. S. Geological Survey*, pp. 228 to 265 (1882).

the C. & C. shaft connection and the "hot winze," the air was practically dead, and in June the average was 100.3°; for July it was 101.6°. On July 17, I measured a temperature of 102° at this point. No. 3 station, at the head of the "hot winze," delivering hot air from the workings of the Ophir mine below this level, gave an average June temperature of 102.8°, and for July, 103.3°. No. 4 station, at the connection with the Union shaft, showed an average temperature of 95.2° for June and 96.3° for July. The lowering of the temperature at No. 4 station was due to the cool air from the Union shaft leaking through a canvas curtain and joining the hot air at both the north lateral and the connection to the Union shaft. At stations Nos. 3 and 4 the temperature was measured in a swiftly-moving air-current.

In the Sutro tunnel, south lateral, no temperature-records were available, but observations made by me showed a temperature of 108.9° in the vicinity of the Julia shaft, Dec. 13, 1908; in the Ward shaft connection, 101.1°; at the junction of the Ward shaft connection and the south lateral, 86°; at the connection of the Sutro tunnel and south lateral, 100°; west of the Combination partition, 94.6°, and east of the partition, 93.7°. A temperature has been noted[3] of 105° for Apr. 15, 1905, in the Sutro tunnel, east of the Combination partition, and during the months of July and August, 1908, this temperature is reported to have prevailed.

The Sutro tunnel is an incast and, as a consequence, the temperatures vary from a minimum at the portal to a maximum at the Combination partition. This variation was measured, and the results are shown in Table II. The tabulated results are plotted in Fig. 2.

An air-temperature of 46° was raised to 95° in passing a distance of 18,000 ft. The air-temperature in the Sutro tunnel varies with the temperature of the outside air, the temperature of the water passing through, and the rock-temperature. The most important factor is the temperature of the water pumped from the C. & C. and the Ward shafts. This water is passed through both the north and south laterals in 24-in. wooden-stave pipes, and at a point 1,000 ft. east of the Combination partition (approximately 18,000 ft. from the portal) the pipe terminates

[3] Leon M. Hall and Frederic W. Bishop, *Report to the Comstock Pumping Association* (May 25, 1908).

TABLE II.—*Variations of Temperature in Sutro Tunnel.*

Date.	Distance from Portal.	Temperature of Water	Temperature of Air.	Relative Humidity.	Notes.
	Feet.	Deg F.	Deg. F.	Per Cent.	
Nov. 22, 1908	Combination partition.	......	94.5	98.5	
Nov. 22, 1908.	18,000	......	94 4	98.2	Near Combination partition.
Nov. 22, 1908.	17,000	95.0	95.0	97.2	End of wood pipe.
Nov. 22, 1908.	13,000	......	95.0	93.7	
Nov. 22, 1908.	10,000	93.7	91.9	98.0	Prospect switch.
Nov. 22, 1908	8,000	92.8	85.2	92	
Nov. 22, 1908	6,800	90 7	73.0	83	Beginning of 30-in. pipe.
Nov. 22, 1908.	5,200	......	71.6	74	
Nov. 22, 1908.	0	......	46.0	59	
Dec. 13, 1908.	E. of Combination partition.	......	93.74	96.4	
Dec. 13, 1908	16,400	92 5	93.2	100	
Dec. 13, 1908	15,000	94.1	93 56	97.2	

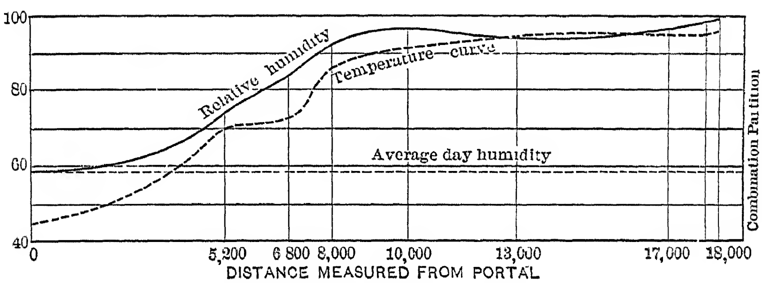

FIG. 2.—CURVES SHOWING TEMPERATURE AND RELATIVE HUMIDITY VARIATION IN THE SUTRO TUNNEL.

and the water flows out, occupying about two-thirds and in some cases the full width of the tunnel. From the portal a 30-in. stave pipe has been constructed a distance of 7,000 ft., and between this point and the former, or a distance of 11,000 ft., the air flows over the surface of the water. This exposure not only heats the air but also saturates it with water-vapor. The close coincidence between the air-temperature and the water-temperature should be noted.

The high temperature in the south lateral near the Julia shaft is due to rock-temperature more than to water, since in the south lateral the water is carried in a wooden pipe. The temperature-measurements in the north and south laterals are shown in Table III.

TABLE III.—*Temperature-Measurements in North and South Laterals of Sutro Tunnel.*

Date	Distance.	Temperature of Water.	Temperature of Air.	Relative Humidity	Notes.
		Deg. F.	Deg. F.	Per Cent	
Dec. 13, 1908.	Surface, Ward shaft.....	.....	40.1	39.6	7.45 a.m.
Dec. 13, 1908.	1,600-level station........	......	88.7	65.5	
Dec. 13, 1908.	Half to south lateral.....	.....	101.1	63.5	
Dec. 13, 1908.	Junction south lateral and Ward connection.	......	87 8	80.4	
Dec. 13, 1908.	South lateral south from charge station.	......	86	84.6	
Dec. 13, 1908.	2,300 ft. from tunnel......	.....	96.8	64.4	
Dec. 13, 1908.	1,550 ft. from tunnel......	.....	105.8	54.4	
Dec. 13, 1908.	400 ft. from tunnel	......	108.86	55.9	
Dec. 13, 1908.	100 ft. from tunnel	......	104 0	63.8	
Dec. 13, 1908.	Sutro tunnel at junction of south lateral.	.....	100	73.7	
Dec. 13, 1908.	Sutro tunnel 100 ft. west of north lateral.	......	87.8	91.4	
Dec. 13, 1908.	75 ft. west of Combination partition	......	94.6	83.9	
Oct. 11, 1908.	Outlet of air-pipe near winze, north lateral.	......	97 0	46.5	
Oct. 11, 1908.	Cross-cut to C. & C. shaft.....................	95 9	93.2	95.0	
Oct. 11, 1908.	Just before reaching Mint shaft.............	......	91.4	95.0	
Oct. 11, 1908	Junction of Sutro tunnel and N. lateral....	......	90 9	94.0	

3. *Water-Temperatures in Sutro Tunnel.*—Opportunities for measuring the water-temperature are few and far between. On Oct. 11, 1908, the water discharged from the C. & C. shaft into the tunnel gave a temperature of 95.9° F.; on Dec. 12 the water from the Ward shaft, close to the Ward, gave a temperature of 141.8°. On the same day, at a point 15,000 ft. distant from the Sutro-tunnel portal, the water-temperature measured 94.1°. The water-flow from the C. & C. shaft amounts to from 4,000 to 4,500 gal., and from the Ward, from 600 to 800 gal. per min. In addition, all the water from the ground above the Sutro tunnel drains into the laterals and discharges into the Sutro tunnel. The average temperature of the water flowing into the Sutro tunnel at the Combination partition is between 94° and 95°. Miners say that sometimes this temperature is exceeded.

4. *Temperatures of Mine-Workings.*—The temperatures in the Ophir workings have been observed for some time, and from

TABLE IV.—*Temperature-Measurements in Different Parts of the Ophir Mine.*

Date, 1907	2,250 Cross cut Station	2,200 Snow-sheds	2,200 Fan	2,100 Ophir.	2,100 N. Stopes	2,200 Station	2,200 Raise
	Deg. F.	Deg. F.	Deg. F.	Deg. F	Deg F	Deg F.	Deg. F.
July 13	75	104	98	98	98	100	111
July 16	68	104	97	98	97	98	110
July 22	78	104	97	97	98	100	111
July 29	72	105	98	98	98	100	110
Aug. 7	72	110	98	98	98	99	111
Aug. 12	68	106	98	98	97	102	111
Aug. 20	66	108	97	98	97	105	113
Aug. 30	69	109	98	98	96	105	112
Sept. 12	66	104	97	98	97	101	112
Sept. 26	68	102	97	96	96	100	111
Oct. 3	62	104	96	97	98	100	110
Oct. 14	70	105	99	98	100	102	112
Oct. 24	64	103	98	101	102	102	110
Oct. 30	62	101	97	100	102	102	109
Nov. 9	58	97	94	97	100	101	110
Nov. 20	54	96	92	96	100	98	108
Nov. 25	55	100	87	95	100	98	108
Dec. 2	57	102	91	97	100	98	108
Dec. 9	57	109	92	99	100	99	109
Dec. 21	56	107	91	96	100	101	109
Dec. 26	57	109	92	96	100	101	108
Jan. 1, 1908	55	106	90	95	100	100	108
Jan. 15, 1908	60	107	91	95	100	100	108
Feb. 1, 1908	54	107	89	94	100	100	108
Feb. 18, 1908	56	112	93	91	101	102	108
Mar. 5, 1908	56	105	92	94	100	102	108
Mar. 31, 1908	57	108	93	96	100	104	108

TABLE V.—*Temperature-Measurements in Different Parts of Ophir Mine and on Different Dates.*

Date, 1908.	2,250 Cross-cut Station.	2,250 Drift.	2,350 Drift.	2,100 N Drift.	2,100 Stopes.	2,200 N E. Drift.	2,200 Stopes.	2,300 Winze.
	Deg. F.	Deg. F.	Deg. F	Deg. F.	Deg. F.	Deg F.	Deg F.	Deg. F.
Apr. 27	63	106	130	104	103	104	105	106
May 20	64	112	112	101	102	101	104	103
May 23	64	114	129	101	102	101	104	102
May 28	65	112	128	100	101	100	102	102
June 1	65	112	126	100	101	101	103	101
June 5	62	110	127	100	101	99	101	101
June 6	62	108	126	100	101	99	101	100
July 20	74	100	102	101	104	104	105	102
July 23	75	100	101	101	102	100	102	100
July 25	75	104	104	102	103	101	102	100
July 30	77	105	110	102	102	105	103	100
July 31	76	101	110	102	105	104	104	101
Aug. 1	78	102	112	102	103	104	104	102
Aug. 3	76	100	110	101	103	104	104	102
Aug. 4	75	106	112	101	104	104	104	101
Aug. 5	74	100	112	101	102	102	102	101
Aug 3, 6, 7	72	100	112	103	102	102	102	102

the daily reports Tables IV. and V. have been compiled. Only a few readings were tabulated, since but very little variation in the temperature takes place from day to day.

From Tables IV. and V. the following averages have been calculated for workings below the Sutro tunnel:

Working-Place.	Average from all Tables.	Average Maximum from all Tables.	Average Minimum from all Tables
	Degrees F.	Degrees F	Degrees F.
2,250 station, C. & C.	67	73.3	62.7
2,200 station, Ophir winze	101.2	102.1	100.6
2,150 drift	99.2	101.6	95.3
2,250 drift	105.6	110.5	101.8
2,350 drift	116.9	125.4	108.5
2,100 stopes	101.2	103.0	99.0
2,200 stopes	103.0	103.2	102.8
2,200 raise	109.7	109.7	....

The temperatures given for the 2,250 and other stations of the C. & C. show the temperatures of the incoming air-currents on the different levels. The measurements in the drifts give the temperatures after the air has been heated to a maximum by contact with the walls of the drifts. A particularly hot zone was passed through by the 2,350 drift close to the 2,350 station of the C. & C. Before connection was made with the Ophir winze, the temperatures were so excessive as to cause the miners much suffering. As soon as a connection was made the temperature dropped about 15°. A progressive increase in temperature with depth is to be noted in the case of the drifts. Few measurements were made at different elevations in the stopes, but the air-current rises in temperature as it ascends in the stope. Confined mine-workings, such as drifts and raises, were usually observed to be the hottest workings, especially where no air-connections exist. Large mine-openings, such as stopes, are usually cooler.

III. The Ventilating-System in General.

A ventilating-current on the Comstock must provide pure air for the miners and a sufficient volume to temper the heat of the underground workings; it must also remove the steam and the gaseous products of blasting. A ventilating-current of sufficient volume to cool the workings enough to be noticeable would meet all the other requirements. The general and consider-

able elevation of temperature along the lode makes it possible to secure air-currents of some volume and velocity by natural ventilation alone. With the exception of the Ward shaft, all of the ventilation is effected by upcast shafts, supplemented by small fans for local distribution of the air in drifts, raises, stopes, and winzes. Three shafts, the Ophir, the Combination, and the Belcher, are used as upcasts; six shafts, the Union, the C. & C., the Mint, the old Yellow Jacket, the Overman, and the Alta, are used as down-casts. The Ward shaft is divided by a brattice; and one compartment (the pump) is used as an upcast, the other two compartments, in which the hoisting is done, serving as a down-cast. A fan is attached to the upcast compartment and exhausts the air.

1. *Ophir Upcast.*—This shaft takes the air from the workings of the Ophir mine, the down-cast air coming principally from the C. & C. shaft. The upcast air is taken from a winze in the north lateral and by a cross-cut to the Ophir incline, and thence to the foot of the Ophir shaft. At the foot of the Ophir shaft a south drift takes air from old stopes and workings of the Ophir mine, and an east cross-cut connects with the Union shaft and takes air from that shaft.

2. *Combination Upcast.*—This shaft draws air from the Sutro tunnel, both east and west of the Combination partition, which is so placed as to split the connection from the Combination shaft to the Sutro tunnel. The down-cast air comes from the Union, Mint, and Alta shafts. The air from the Union crosses the current from the Ophir through a 20-in. galvanized-iron pipe, two wooden partitions keeping the currents from mixing. The air from the Mint shaft is taken through three lines of 15-in. air-pipe, assisted by fans, to the Gould and Curry, the Savage, the Chollar, and the Hale and Norcross workings. It is then returned through the north and south laterals to the Sutro tunnel, and thence to the Combination shaft. The air from the Alta passes through a long connection and into the south lateral near the connection to the Ward shaft. The air east of the Combination partition comes from the Sutro-tunnel portal.

3. *Belcher Upcast.*—This shaft ventilates the Yellow Jacket, the Overman, and the Caledonia mines, and draws air for this purpose down the old Yellow Jacket and the Overman shaft.

The total amount of upcast air from the three shafts and the

Ward measures 235,835 cu. ft. per min., and the amount of down-cast air measures 216,687 cu. ft. per min. The discrepancy between the two amounts is due to the difficulty of making accurate measurement, of rise in temperature, leakage, variation in the velocity of the air-current at different times, and the accession of water-vapor and gases from the mine-workings. The distribution of the air between the shafts is shown by Table VI.

TABLE VI.—*Distribution of Air in Upcasts, Down-Casts, and the Sutro Tunnel.*

Shaft	Number of Compartments.	Area.	Quantity.	Temperature at Collar	Temperature at Level.	Connecting Level at Bottom.	
		Sq. Ft	Cu. Ft. per Min.	Deg. F.	Deg. F.		
Combination......	4	129	51,651	87.6	95.2	1,600	
Ophir..............	4	94	58,454	88.9	98.8	1,600	
Belcher..........	3	60	31,706	73.9	87.3	1,100	Yellow Jacket
Ward..............	1	33	94,024	67.5	104	2,475	
Total upcast air.	12	316	235,835				
C. & C...........	3	77.3	28,941				
Union............	4	126	13,814				
Yellow Jacket...	3	64.2	10,665				
Alta...............	2	43.3	13,364				
Mint...............	$1\frac{1}{2}$	30	7,500				
Overman on 900-ft level..	3	60	5,981				
Unaccounted for in Belcher	....	....	15,060				
Sutro tunnel......	1	63	27,339				
Ward shaft........	2	55	94,024				
Total down-cast air.......	$19\frac{1}{2}$	518.8	216,687				

From Table VI. the average velocity in feet per minute has been calculated as follows:

	Cu. Ft. Per Min.
Whole upcast area, ...	746.3
Whole down-cast area, ...	417.6
Whole upcast area, excluding Ward shaft, ...	501.1
Whole down-cast area, excluding Ward shaft, ...	264.4

An interesting comparison with past conditions is afforded by the data given by Church.[4] Table VII. is quoted from his treatise.

[4] *The Comstock Lode: Its Formation and History*, p. 18 (1879).

TABLE VII.—*Distribution of Upcast Air, July 2, 1877.*

Shaft.	Quantity.	Temperature of Up cast at Top of Shaft.
	Cu. Ft. per Min.	Degrees F.
Utah	4,000	.....
Sierra Nevada	7,700	76
C. & C.	21,600	84
Con. Virginia	48,750	89
Gould and Curry	12,000	.....
Savage	58,500	100
Chollar-Potosi	18,000	77
Bullion	10,080	89.5
Imperial Consolidated	28,800	95
Belcher	52,200	89
Overman	27,000	93
Total upcast air	288,630	Aver. temp.....88.05

Six shafts served as down-casts, but were not named. Church estimates the above quantity of air as a minimum, and states that probably 300,000 cu. ft. represents the average amount of upcast air, and 10,000 cu. ft. per min. as the quantity of air due to the air-compressors in use. The outside temperature for the day upon which the measurements were made is given as 73° F.

The average amount of air per upcast, from Table VII., is 26,240 cu. ft. per min., and temperatures at the top of the shaft average 88.05°. Excluding the Ward shaft, Table VI. shows an average of 47,270 cu. ft. per min. per shaft, and a top-temperature average of 83.4°. The figures indicate a somewhat greater effect for the upcast shafts at the present time than in the past. The lower average temperature shows the effect of the restricted workings. The fewer upcast shafts and the restricted workings of the present time would indicate, on the whole, a greater resistance to the movement of the air than at the time Church made his studies. The greater effectiveness of the present upcast shafts could be explained by the more liberal use of small fans. No detailed data are available concerning the use of small fans during the early periods of the Comstock. Of the shafts mentioned by Church, only the C. & C., Belcher, and Overman are open, and of these the C. & C. and Overman are used as down-casts, while the Belcher is an upcast at the present time.

Fig. 3 shows the relation of the underground workings, the principal upcast and down-cast shafts.

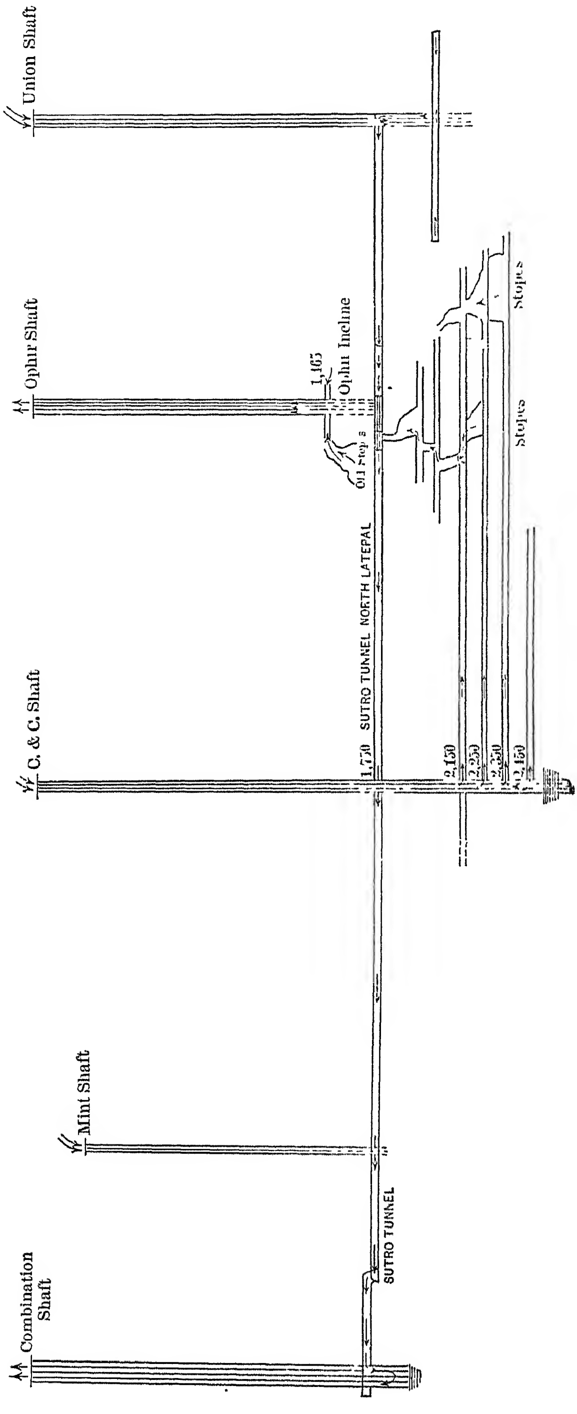

FIG. 3.—LONGITUDINAL SECTION, SHOWING WORKINGS AND SHAFTS OF NORTH-END MINES.

IV. The Ventilating-System in Detail.

1. *Union Shaft.*—The down-cast air is taken to the 2,000-ft. level, and thence by means of a 15-in. air-pipe to a fan 600 ft. distant in the east cross-cut. It is then forced to the face of the NW. cross-cut, and also to the face of the south drift, distances respectively of 500 and 1,000 ft. The discharges are respectively 1,042 and 796 cu. ft. per min. The combined discharge is 1,838 cu. ft. per min. With an inflow at the Union shaft of 4,287 cu. ft. per min., the delivery is 42.9 per cent. of inflow. The air returns through the east cross-cut to the shaft and then passes up through the pump-compartment, which is bratticed off, to the Sutro-tunnel level, and then through a long connection to the south lateral and also to the connection leading to the Ophir shaft. The north lateral takes 4,086 cu. ft. per min., and the remainder, 9,728 cu. ft. per min., finds its way into the Ophir upcast. The average temperature on the 2,000-ft. level is 98.3° F., and on thé Sutro-tunnel level it is 76.3°. This is due to the fact that a comparatively large proportion of the down-cast air leaks past the air-pipe on the 2,000-ft. level and goes up the pump-compartment, cooling the air coming from the Union workings. The portion of air passing to the Combination shaft traverses 12,061 ft. from surface to surface; the portion passing up the Ophir moves 7,680 ft. Fig. 4 shows in plan the distribution of the air on the Sutro-tunnel level, and Table VIII. gives the measurements made on the same level.

2. *The Ophir Mine.*—The down-cast air in the C. & C. shaft splits into four parts—one split on the 2,150-ft. level, an air-current of 7,024 cu. ft. per min.; one on the 2,250-ft. level, an air-current of 14,521; one on the 2,350-ft. level, of 11,667, and one on the 2,350-ft. level, which was not measured.

On the 2,150-ft. level the air is taken through the drift to a point 860 ft. from the shaft, then picked up by a fan and forced to the Ophir winze and down through that connection to the stopes upon the 2,200-ft. level of the Ophir. The fan takes 5,389 cu. ft. per min., and the difference, 1,635 cu. ft., leaks past a canvas curtain and joins the upcast air.

On the 2,250-ft. level part of the air is taken by a fan at the station and forced down to the 2,350-ft. level to ventilate the

TABLE VIII.—*Measurements of Air, Temperatures, and Relative Humidities at the Sutro-Tunnel Level.*

Section.	Date, 1908.	Velocity.	Area Section.	Quantity.	Temperature	Relative Humidity	Notes.
		Ft per Min.	Sq. Ft.	Cu. Ft. per Min	Deg. F.	Per Cent.	
F.	Dec. 28.	228	63	13,364	84 2	86 2	Air from Alta shaft.
F.	Dec. 13	161	63	10,134	86.0	84.6	Air from Alta shaft.
G. & C.	Nov. 15	1,159	1.23	1,426	65.2	71.0	To Gould and Curry through 15-in. pipe.
G. S.	Nov. 15.	904	2.66	2,404	65.2	71.0	To Savage through 15-in. pipe.
H. & N.	Nov. 15	2,452	1.39	3,408	65.2	71.0	To Hale and Norcross through 15-in. pipe.
G. & C.	Dec. 28.	1,122	1.23	1,380	63.1	77 8	To Hale and Norcross through 15-in. pipe.
G. S.	Dec. 28.	886	2.66	2,357	63.1	77.8	To Hale and Norcross through 15-in. pipe.
H. & N.	Dec. 28.	2,706	1.39	3,761	63.1	77 8	To Hale and Norcross through 15-in. pipe.
T.	Dec. 13.	1,035	8.1	8,221	87.8	91.4	Through door.
Winze.	July 15.	980	20.0	19,600	107.6	100—	Mine air.
J.	July 16.	650	39.0	25,336	105.4	100—	Curtain at N. closed.
J.	July 16.	433	39.0	16,454	105.4	....	Curtain at N. open.
J.	July 17.	600	39.0	23,400	104	97	Curtain at N. closed.
J.	July 17.	329	39.0	12,831	104	....	Curtain at N. open.
J.	Oct. 11.	567	54.5	30,901	93.2	95	Partition at N.
J.	Dec. 12.	874	39.0	30,186	99.0	81.2	Partition at N.
J.	Dec. 29.	940	35.0	32,760	100 4	76.4	Partition at N.
K.	July 15.	1,060	29.7	31,482	101.3	100—	Curtain at N. closed.
K.	July 16.	1,108	29.7	32,907	100.4	100—	Curtain at N. closed.
K.	July 17.	730	35	25,550	98.6	97.9	Curtain at N. closed.
K.	July 17.	809	35	28,315	95.0	....	Curtain at N. open.
M.	July 16.	464	35	16,440	105.8	....	Air from old stopes.
M.	July 17.	330	35	11,550	104.9	....	Air from old stopes.
L.	July 16.	210	20	4,200	93.2	....	Air from Union shaft.
L.	July 17.	145	20	2,900	95.9	....	Air from Union shaft.
R.	July 15.	190	56	10,640	91.4.	....	
R.	July 16.	190	35	6,650	87.8	....	Curtain closed at N.
R.	July 16	376	35	13,160	87.8	....	Curtain open at N.
R.	July 17.	120	35	4,200	88	73.4	Curtain closed at N.
R.	July 17.	221	35	7,735	85.8	... .	Curtain open at N.
R.	Dec. 12.	3,291	0.4	1,316	86.4	66.8	Air-pipe in partition.
N.	Dec. 12.	364	22.4	8,143	82.4	72.4	Curtain removed.
O.	Dec 12.	278	14.72	4,086	82.4	72.4	Air going to I.
P.	Dec. 12.	109	31.00	3,379	82.4	72.4	
Q.	Dec. 12.	478	28.85	13,814	82.4	70.3	
H.	Oct. 11.	1,632	2.07	3,378	97	46.5	20-in. pipe.
H.	Dec. 29.	1,750	2.07	3,622	94	48.0	20-in. pipe.
I.	Oct. 11.	1,750	2.00	3,500	93.6	49.0	20-in. pipe.

east drift; the remainder, amounting to 14,521 cu. ft. per min., passes into the drift until it reaches a stope, where two fans pick it up, one forcing the air past the stope into the Consolidated Virginia workings, the other forcing the remaining air down through the Ophir winze to the 2,200-ft. level of the Ophir mine. The Consolidated Virginia fan (No. 3) takes 3,323 cu. ft. per min., while the Ophir fan (No. 2) takes 11,566 cu. ft. per min. Beyond the stope another fan (No. 1) takes air from the Ophir-winze station and forces it on to the stopes north of the Ophir winze. This air rises up through the stopes and event-

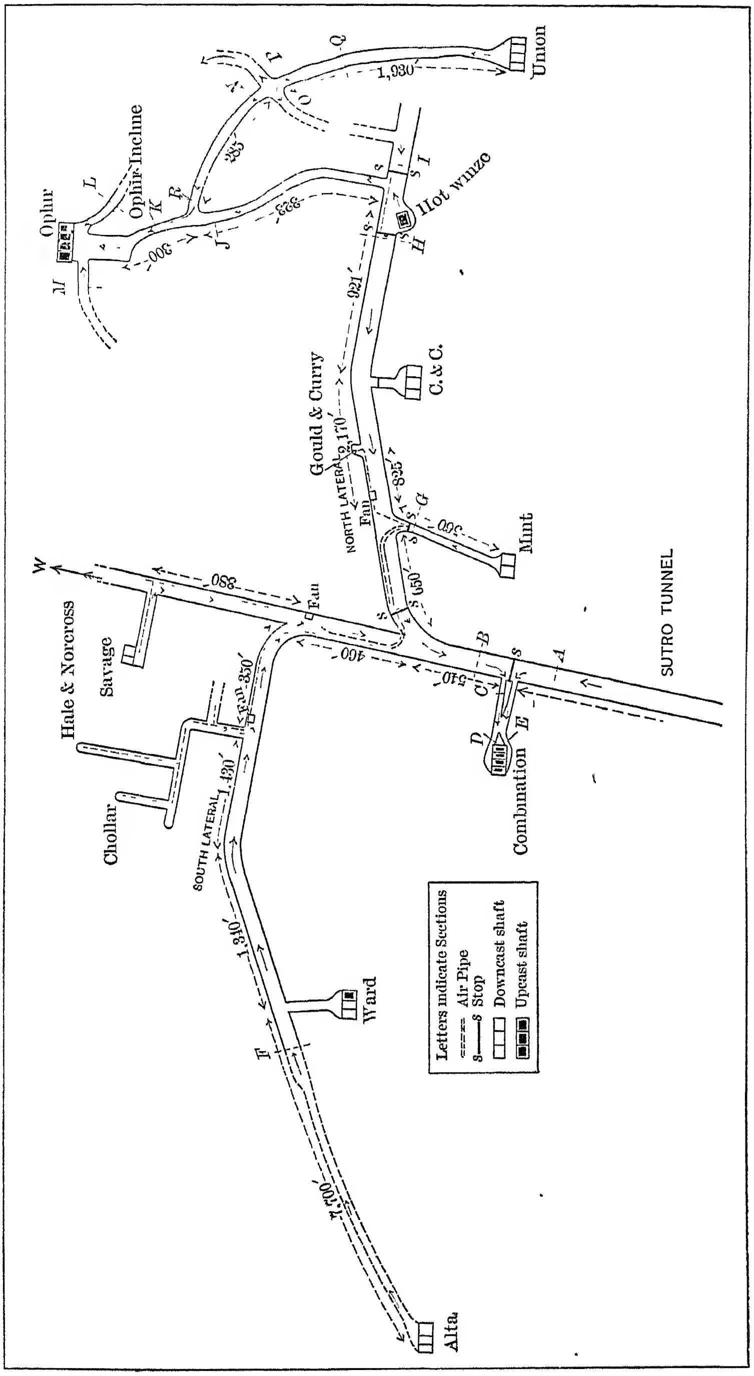

Fig. 4.—Plan Showing Distribution of Air on Sutro-Tunnel Level.

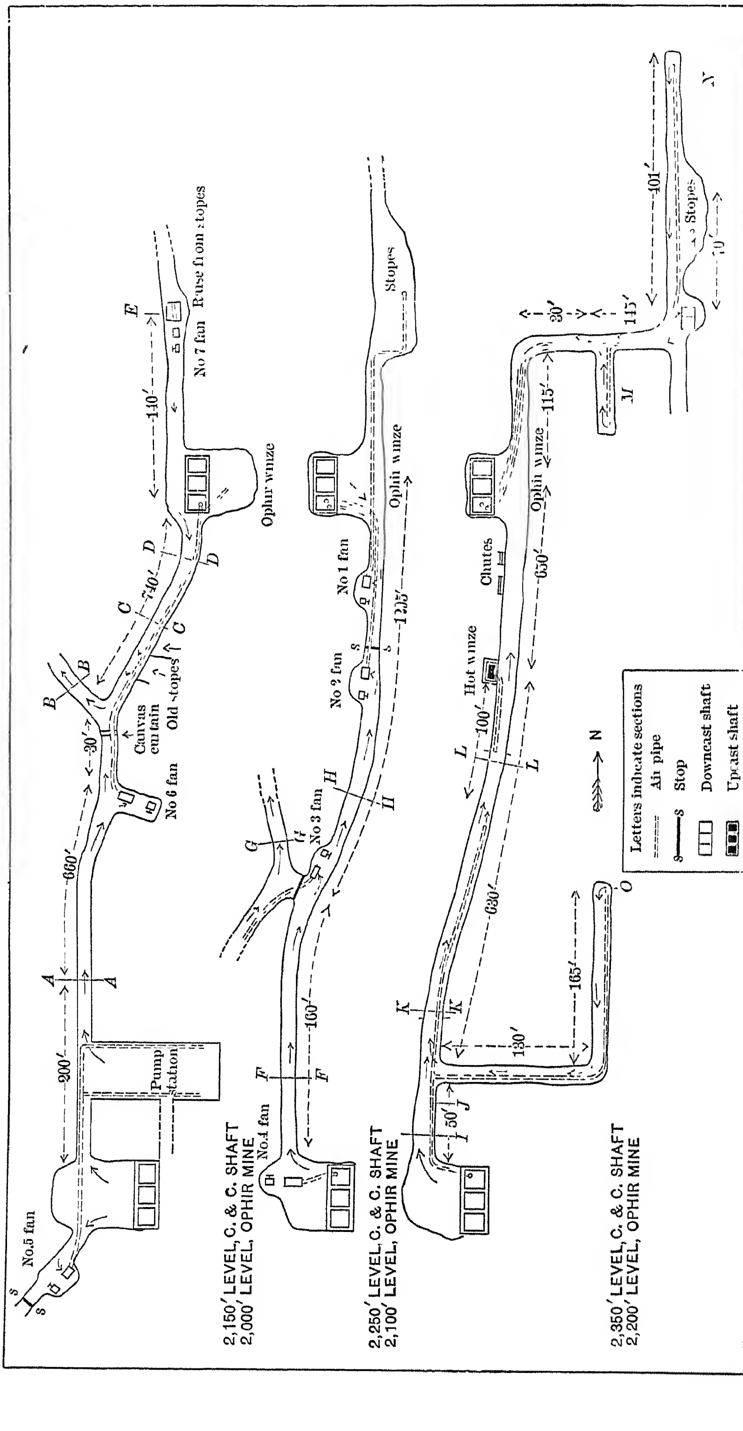

FIG. 5.—PLAN OF 2,000-, 2,100-, AND 2,200-FT. LEVELS OF THE OPHIR MINE.

ually joins the air in the "hot winze" of the north lateral. The fan takes 3,791 cu. ft. per minute.

The air on the 2,350-ft. level passes through the main drift, while the air from the fan on the 2,250-ft. level passes into a cross-cut and a drift east of the main drift, returns, and joins the air passing through the main drift towards the Ophir winze. The high temperatures, noted previously, were taken in this drift before the connection was made. The average temperature before connection was 125.4°, and some months later the temperature measured 100.4°. The lowering in temperature was due to the increased volume of air and to the boarding-up (snow-shedding) of the sides and top of the drift, which very effectually shut off hot water, steam, and much of the heat radiated from the rock-surfaces.

The 2,450-ft. level is ventilated by a No. 8 cast-iron American blower. The air is taken through an inlet-pipe extending towards the south compartment of the shaft and forced to the working-face in the east cross-cut, returning through the cross-cut. The fan was not in operation at the time of my visit. On account of the danger of flooding this level the fan was driven by a compressed-air engine.

The total quantity of air in the three main splits amounts to 33,212 cu. ft. per min., and this compares closely to the 32,760 cu. ft. measured on the same day on the Sutro-tunnel level. Fig. 5 shows the plans of the three levels of the Ophir mine, with the sections at which velocity-measurements were made. The measurements are given in Table IX.

3. *Gould and Curry Mine.*—At the Sutro tunnel a west cross-cut has been started. This is ventilated by an air-pipe and fan which takes air from the Mint shaft. The inflowing air at the shaft measured 1,122 cu. ft. per min. (1,159 cu. ft., Nov. 15). A 15-in. air-pipe takes this air to the fan, and from the fan an 11 in. pipe connects with the cross-cut, distant 189 ft. The air discharged measured 989 cu. ft., or 88.1 per cent. of the inflow. The fan was not in use.

4. *Savage Mine.*—An east cross-cut is being run in the Savage mine from a raise 75 ft. above the Sutro tunnel, which also receives air from the Mint shaft. A 15-in. air-pipe takes 2,404 cu. ft. per min. to a fan 1,110 ft. distant in the Sutro tunnel, and the fan discharges through a 15-in. pipe to the top of the

TABLE IX.—*Air-Measurements at Different Sections in Ophir Mine.*

Date, 1908.	Level.	Section.	Area.	Velocity.	Quantity.	Temperature.	Relative Humidity.	Notes.
			Sq.Ft.	Ft. per Min.	Cu. Ft. per Min.	Deg. F.	Per Cent.	
July 15.	2,000	A.	29.75	240	7,140	97.2	36.5	
Dec. 29.	2,000	A.	31.50	223	7,024	87.4	34.2	
July 15.	2,000	No. 6 fan.	1.77	2,766	4,896	....	......	Velocity low.
Dec. 29.	2,000	No. 6 fan.	1.77	3,045	5,389	98.6	37.5	Inlet of fan.
July 15.	2,000	B.	29.75	480	14,280	116.6	100—	Air to upraise to Sutro tunnel.
Dec. 29.	2,000	B.	23.22	620	14,396	108.1	86.7	Air to upraise to Sutro tunnel.
July 15.	2,000	C.	28.05	175	4,900	112.3	45.4	
Dec. 29.	2,000	C.	26.00	156	4,056	105.8	58.8	
Dec. 29.	2,000	D.	25.6	434	11,110	107.6	59.1	
July 15.	2,000	E.	20	225	3,375	100	98.5	
July 15.	2,100	F.	29.75	397	11,810	76.1	95.4	
Dec. 29.	2,100	F.	31.5	461	14,521	64.8	93.3	
July 15.	2,100	G.	26	320	8,320	96	89	
Dec. 29.	2,100	No. 3 fan.	2.82	1,182	3,323	....	......	
July 15.	2,100	H.	29.75	280	8,330	82.4	79.8	
Dec. 29.	2,100	H.	30.9	284	8,776	90.1	55.3	
Dec. 29.	2,100	No. 2 fan.	7.01	1,650	11,566	90.1	55.3	
Dec. 29.	2,100	No. 1 fan.	1.67	2,270	3,791	100.4	49.7	Temperature taken at Ophir-winze station.
July 15.	2,200	I.	28.25	220	6,140	89.6	96.4	Connection made.
July 15.	2,200	J.	1.22	3,492	4,160	82.4	......	End of 15-in. air-pipe; air from No. 4 fan.
Dec. 29.	2,200	K.	28.25	413	11,667	84.2	88.4	
Dec. 29.	2,200	L.	....	....	....	98.6	98.8	End of snow-sheds.
Dec. 29.	2,200	Hot winze	....	....	....	102.7	93.8	
Dec. 29.	2,200	....	....	....	....	104	98.8	50 ft. south of Ophir winze
Dec. 29.	2,200	Ophir winze.	....	....	....	100.4	45.8	
Dec. 29.	2,200	O.	1	1,996	1,996	89.6	40.6	Air from pipe, No. 4 fan.
Dec. 29.	2,200	M.	1.22	606	739	102.2	48.5	Air from pipe, No. 1 fan.
Dec. 29.	2,200	N.	1.11	490	544	104.5	36.5	Air from pipe, No. 1 fan.
Dec. 29.	2,200	....	....	....	....	93.6	72.6	Stopes between 2,100- and 2,200-ft. levels.

raise and then by an 11-in. pipe to the working-face, 180 ft. distant from the top of the raise. The discharge measures 894 cu. ft. per min., or 37.1 per cent. of the inflow. The section of 11-in. pipe caused the most leakage. The fan was in operation at the time.

5. *Hale and Norcross Mine.*—The Hale and Norcross receives air from the Mint shaft. A 15-in. air-pipe leads to the H. & N. connection in the south lateral, 1,460 ft., and from this point a fan takes the air and forces it to the H. & N. and Chollar cross-cuts, distant 325 ft.; 11-in. branch-pipes take the air to the working-faces, 190 ft. to the H. & N. and 122 ft. to the Chollar. The H. & N. discharge measures 941 and the Chollar 934 cu. ft. per min. The combined discharge measures 1,875 and the inflow at the Mint 3,761 cu. ft. per min. The discharge is 49.8 per cent. of inflow. The measurements of both the Chollar and the H. & N. are given in Tables X. and XI. and Fig. 6.

TABLE X.—*Air-Measurements in Hale and Norcross and Chollar Mines.*

Point.	Velocity.	Area.	Quantity.	Temperature.	Relative Humidity	Notes
	Ft per Min.	Sq. Ft	Cu Ft per Min	Deg. F	Per Cent.	
Mint Shaft.	2,452	1.39	3,408	65.2	71	Nov. 15, 1908. Air going into H & N. air-pipe.
Mint Shaft.	2,706	1.39	3,761	63.1	77.8	Dec. 28, 1908. Air going into H & N. air-pipe.
B.	1,416	0.66	934	104.5	23.8	Nov. 15, discharge of 11-in. pipe.
B.	....	....	....	105.1	61.4	Dec. 28, discharge at J, 78 ft. from B.
J.	1,093	0.66	721	106.2	22.0	Dec. 28, discharge of 11-in. pipe.
C.	....	....	....	97.5	69.0	Nov. 15.
C.	....	....	....	104.0	51.2	Dec. 28.
E.	1,426	0.66	941	106.1	21.8	Nov. 15, discharge of 11-in. pipe.
E.	....	....	....	107.6	32.2	Face of drift 12 ft. from end of pipe, Nov. 15.
E.	1,043	0.66	688	109 4	20.0	Discharge of 11-in. pipe, Dec. 28.
E.	....	....	....	109.6	29.8	Face of drift 25 ft. from end of pipe, Dec. 28.
D.	....	....	....	95.5	66.2	Nov. 15.
F.	....	....	108	81.7	51.4	Leakage, 15-in. damper; cooling bench, Nov. 15.
F.	2,410	1.39	3,350	104.0	24.8	Damper on split closed; F open, Dec. 28.
F.	1,360	1.39	1,890	104.0	24.8	Both dampers open, Dec. 28.
F.	8	1.39	....	104 0	24.8	Damper at F closed, Dec. 28.
G.	9	....	....	98 1	64.4	Nov. 15.
G.	....	....	....	100.4	62.5	Dec. 28.
H.	....	....	....	105.8	66.6	Nov. 15.
H.	....	....	....	108.1	62.7	Dec. 28.
A.	....	....	....	105.5	58.5	Nov. 15.
A.	....	....	....	104.0	63.8	Dec. 13.
H.	....	....	....	108 86	55.9	Dec. 13.

TABLE XI.—*Inflow and Discharge of Air at Hale and Norcross and Chollar Mines.*

Date, 1908.	Inflow.	Discharge.	Air Lost.	Quantity Delivered.	Distance.
	Cu. Ft. per Min.	Cu. Ft per Min.	Cu. Ft. per Min.	Per Cent.	Ft.
Nov. 15......	3,408	1,983	1,425	58	2,137
Dec. 28.......	3,761	1,409	2,352	37.4	2,155
Dec. 28.......	3,761	3,350	411	89	1,730

6. *Sutro Tunnel.*—The Sutro tunnel is an incast air-way and is the longest air-way on the Comstock. A plan of the tunnel at the Combination partition is shown in Fig. 7. The air enters at the portal and passes to the Combination partition,

a distance of 18,700 ft.; then through the connection to the Combination shaft, a distance of 400 ft., and then to the surface, a distance of 1,600 ft. vertically. The total distance is 20,700 ft. This Combination shaft also serves to draw air from the workings west of the Combination partition. The average quantity of air from the east is 27,339 cu. ft. per min. (average of four observations at different times); from the west it is 29,961 cu. ft. per min. (average of four observations at different times); and the total average passing to the Combina-

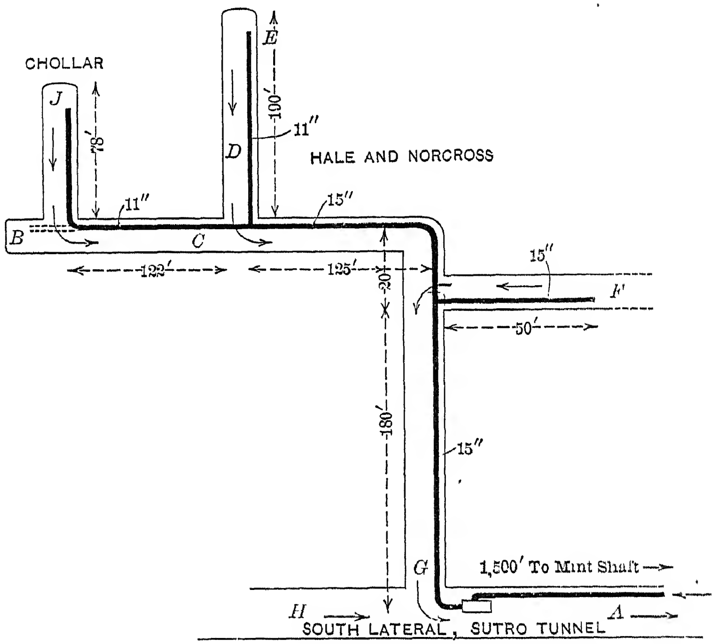

FIG. 6.—PLAN OF CHOLLAR AND HALE AND NORCROSS WORKINGS.

tion from both directions measures 57,300 cu. ft. per min. The observations made at the Combination partition are tabulated in Table XII.

An interesting comparison is afforded by a measurement made in the Sutro tunnel, Mar. 7, 1909. The Sutro tunnel just west of the Combination partition was completely blocked by a cave, and the Combination shaft drew air only from the east side. The quantity of air measured 30,124 cu. ft. per min.; temperature, 89.6° F.; relative humidity, 100 per cent. The temperature at the foot of the shaft measured 92.3°; relative

TABLE XII.—*Observations on Air-Currents at the Combination Partition.*

Date, 1908.	Section.	Velocity.	Area.	Quantity.	Temperature.	Relative Humidity	Barometer	Time
		Ft. per Min.	Sq Ft	Cu Ft. per Min.	Deg. F.	Per Cent.	In.	
Oct. 11...............	D.	730	35.72	26,075	94.3	96	.... ..	11.20 a.m.
Oct. 11...............	E.	850	27.58	23,443	95.4	95.4	.. ....	11.20 a.m.
Oct. 11........	C.	1,154	18.9	21,810	95.54	85.7	....	11.20 a.m.
Oct. 11.......... ..	A.	492	48.5	23,813	97.52	93.0	....	11.20 a.m.
Nov. 15...	A.	523	48.5	26,500	95.0	100	25.4	....
Nov. 22.............	A.	631	48.5	32,181	94.5	98.4	25.15	9.00 a.m.
Dec. 13	A.	604	48.5	30,200	93.7	96.4	25.10	10.00 a.m.
Dec. 28.............	A.	480	48.5	24,000	94.1	96.4	25.35	11.00 a.m.
Nov. 15.............	B.	627	46.3	29,307	95.5	85	25.4	
Nov. 22...........	B.	732	46.3	33,891	95.2	83.9	25.1	9.00 a.m.
Dec. 13.........., ..	B.	628	46.3	29,076	94.64	83.9	25.15	11.10 a.m.
Dec. 28.............	B.	594	46.3	27,572	94.50	83	25.39	11.00 a.m.

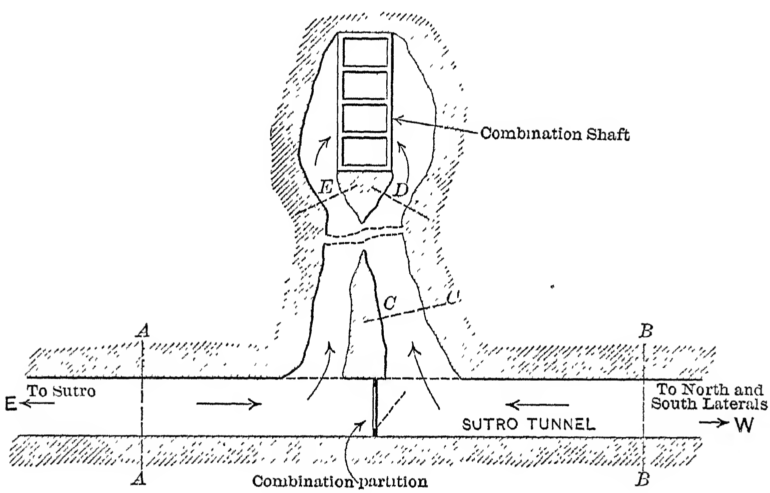

FIG. 7—PLAN OF SUTRO TUNNEL AT COMBINATION PARTITION.

humidity, 96.5 per cent.; barometer, 25.15 in. A measurement was taken on the same date, within 2 hr. of the measurement in the tunnel, at the collar of the Combination shaft, and 32,719 cu. ft. per min. at a temperature of 72°; relative humidity, 100 per cent.; and barometric height, 23.85 in., was the result. These figures show a pronounced cooling-effect of the shaft with a smaller volume of air passing, and this would de-

crease the efficiency of the shaft as an upcast. Instead of obtaining a greater volume of air by turning the full effect of the upcast shaft upon the tunnel, only a small increase resulted. This is a direct result of the decreased efficiency of the shaft.

A recording-anemometer was placed in the Sutro tunnel close to the Combination partition, and three 24-hr. records taken of the east air. The observed velocities are tabulated in Table XIII., the average velocities in Table XIV., and the quantities of air in Table XV. The anemometer was placed in the position shown in Fig. 8. The ratio between the average velocity for the section and the velocity for the position was determined by measurements with the small anemometer. This ratio was found to be 1.12, and this figure was used in reducing the observed velocities. The area of the section is 48.5 sq. feet.

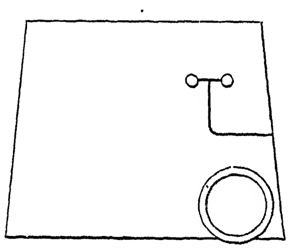

FIG. 8.—SECTION OF SUTRO TUNNEL EAST OF COMBINATION PARTITION, SHOWING POSITION OF ANEMOMETER.

TABLE XIII.—*Velocity-Readings.*

Time.	Nov. 21 to 22, 1908.	Nov. 22 to 23, 1908.	Nov. 23 to 24, 1908.	Time.	Nov. 21 to 22, 1908.	Nov. 22 to 23, 1908.	Nov. 23 to 24, 1908.
	Miles Per Hr.	Miles Per Hr.	Miles Per Hr.		Miles Per Hr.	Miles Per Hr.	Miles Per Hr.
12 to 1 p.m.	....	....	....	1 to 2 a.m.	5.00	4.70	5.60
1 to 2 p.m.	....	3.90	....	2 to 3 a.m.	5.00	4.80	5.50
2 to 3 p.m.	....	3.75	....	3 to 4 a.m.	4.20	4.62	5.90
3 to 4 p.m.	4.25	4.40	4.90	4 to 5 a.m.	4.34	4.75	5.50
4 to 5 p.m.	4.00	4.30	5.40	5 to 6 a.m.	3.90	5.13	5.50
5 to 6 p.m.	4.33	4.20	5.20	6 to 7 a.m.	4.16	4.80	5.40
6 to 7 p.m.	4.30	4.50	5.00	7 to 8 a.m.	4.55	4.80	5.74
7 to 8 p.m.	4.30	4.44	5.00	8 to 9 a.m.	4.62	5.00	5.45
8 to 9 p.m.	4.30	4.64	5.20	9 to 10 a.m.	4.60	4.80	5.40
9 to 10 p.m.	4.47	4.34	5.25	10 to 11 a.m.	4.26	4.20	5.30
10 to 11 p.m.	4.93	4.50	5.55	11 to 12 a.m.	4.50	4.10	5.30
11 to 12 p.m.	5.07	4.55	5.64	12 to 1 p.m.	4.25	4.20	4.60
12 to 1 a.m.	5.13	5.10	5.38	1 to 2 p.m.	....	4.00	....

TABLE XIV.—*Average Velocities.*

Date. Nov., 1908	Average.		Maximum.		Minimum		Time, Maximum.	Time, Minimum.
	Miles Per Hr	Ft. Per Min.	Miles Per Hr.	Ft. Per Min	Miles Per Hr.	Ft. Per Min.		
21 to 22.	4.98	438.3	5.75	505.2	4.37	384.4	12 to 1 a.m.	5 to 6 a.m.
22 to 23.	4.65	409.2	5.75	505.2	4.37	384.4	5 to 6 a m.	1 to 2 a.m.
23 to 24.	5.99	527.1	6.61	581.3	5.16	453.4	3 to 4 a.m.	12 to 1 p.m.
Average..	5.21	458.5	6.04	530.6	4.63	407.4	....	....

The observed velocities have been plotted in Fig. 9.

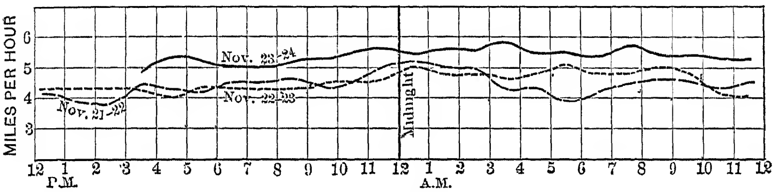

FIG. 9.—CURVES SHOWING RELATION BETWEEN VELOCITY AND TIME OF DAY.

TABLE XV.—*Air in Cubic Feet Per Minute.*

Date.	Average Quantity of Air.	Maximum Quantity of Air.	Minimum Quantity of Air.
	Cubic Feet.	Cubic Feet.	Cubic Feet.
Nov. 21 to 22....	21,257	24,502	18,859
Nov. 22 to 23....	19,846	24,502	18,859
Nov. 23 to 24....	25,564	28,193	21,990
Average...........	22,222	25,732	19,903

The average quantity, as shown in Table XV., and all the velocities, with the exception of one, are lower than the average velocity obtained from the readings in Table XII. Maximum and minimum velocities do not occur at any particular time. The wavy character of the velocity-curves is noteworthy, and is probably due to the fact that at several points in the tunnel are constrictions, often for some little distance, and the added friction produced by these results in more or less of a pumping-action. Undoubtedly, a more delicate recording-instrument would have shown the larger waves to be formed of a multitude of smaller ones. The movement of cars in the tunnel does not appear to have any decided effect upon the curves, although more delicate measuring-instruments might have shown it.

7. *Yellow Jacket Mine.*—Three levels of this mine are open

and operated from the old Yellow Jacket shaft. On the first level, the 900-ft., the air entering measures 3,075 cu. ft. per min. The air passes through a drift and thence down a winze to the 1,000-ft. level. The temperature and relative humidity of this air-current, at a point 500 ft. from the shaft, measured 66.2° F. and 55.6 per cent. respectively. The second, or 1,000-ft. level, receives 3,030 cu. ft. per min. from the shaft. Part of this air is taken by an air-pipe and conducted a distance of 350 ft. to a raise on the foot-wall of the vein, returns, joins the remainder, and then passes down through old stopes to the 1,100-ft. level.

The third, or 1,100-ft. level, is closed by a door in the drift, so as to force the air down the incline, which heads at this level, into an old station which is being reopened. The air escapes up through old stopes to the 1,100-ft. level. All the air passing down the Yellow Jacket shaft is brought together in the south drift, passes through a cross-cut to the old Belcher incline, and thence to the surface through the Belcher shaft. The amount of air passing through the drift measures 10,665 cu. ft. per min. The 900-ft. level of the Overman and Caledonia, corresponding to the 1,100-ft. of the Yellow Jacket, delivers 5,981 cu. ft. per min. to the Belcher shaft. The total quantity of air measured on this level is 16,646 cu. ft. per min., while the average passing up the Belcher is 31,706, leaving unaccounted for 15,060 cu. ft. per min. This comes from the lower levels of the Overman and Caledonia, but no measurement was possible at the time. The Yellow Jacket air at the foot of the Belcher incline gave a temperature of 79.7° and a relative humidity of 68 per cent.; the air coming from the Overman a temperature of 82.4° and a relative humidity of 62.3 per cent.

8. *Ophir and Belcher Upcasts.*—The measurements of both these shafts are to be found in Tables XVI. and XVII.

9. *Combination Shaft.*—The measurements for the Combination shaft are given in detail in Table XVIII. The numbered circles in Fig. 10 indicate the position in which the anemometer was held for each measurement.

Table XIX. has been calculated from the data of Tables XVI., XVII., and XVIII.

Since the cooling of the upcast air directly affects the effi-

TABLE XVI.—*Air-Measurements in Ophir Upcast Shaft.*

Date, 1908.	Pump.	No. 1.	No 2	No 3	Average	Quantity.	Shaft-Temperature.	Relative Humidity, Shaft.	Surface-Temperature.	Relative Humidity, Surface.	Barometer
						Cu Ft per Min.	Deg. F.		Deg F		
July 14, 11 45 a m.	647	699	636	.... .	660	62,040	90 5	100+	73 4	30 45	24.2
July 17, 1 p m ..	532	506	480	. . ...	506	47,564	. .	100+	81 0	16 40	24.2
Aug. 8, 4 p.m. ..	546	583	506	. . ...	523	49,162	93.4	100+	87 8	18.40	24 0
Oct. 18, 2 p.m .	726	405	759	. . .	630	59,220	89	100+	40 0	. .. .	23.8
Nov. 21, 4.30 p m.	716	701	747	. ... .	721	67,774	89	100+	45.0	61.2	23.7
Dec. 22, 5.30 p m.	584	668	706	......	653	61,382	86	100+	40.6	26 5	23.97
Dec. 27, 3 30 p m.	653	652	675	......	660	62,040	86	100+	43.3	50.1	24.0
Average	....	....	....	......	622	58,484	88.9	100+			

TABLE XVII.—*Air-Measurements in Belcher Upcast Shaft.*

Date, 1908.	No 1	No. 2.	No. 3.	Average.	Quantity.	Shaft-Temperature.	Relative Humidity, Shaft.	Surface-Temperature.	Relative Humidity, Surface.	Barometer.
					Cu Ft. per Min.	Deg. F.		Deg. F.		
Oct. 10, 4 p.m ..	620	567	323	595	27,720	. .	100+	. . . .	.....	. .
Nov. 8, 4 p.m.. . .	656	612	525	598	30,611	... ..	100+	.... .	.. ....	. . ..
Nov. 15, 10.45 a.m	760	717	525	667	36,375	74.3	100+	50	....	......
Nov. 22, 9 a.m... .	835	724	405	655	33,915	74.0	100+	50	... .	.. ..
Dec. 18, 3 p.m ...	578	541	586	561	28,714	73	100+	26.6	49.2	24.2
Dec. 30, 11.45 a.m.	692	613	323	543	32,992	74.3	100+	44.6	50.2	23.9
Average....	....	....	... .	603	31,706	73 9				

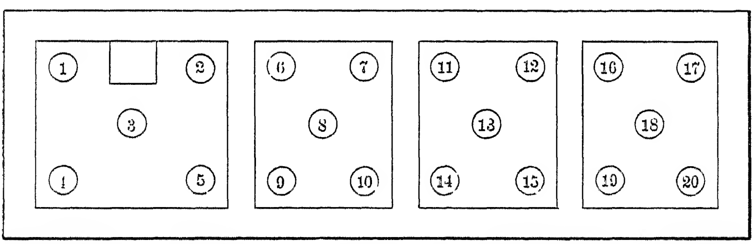

FIG. 10.—SECTION OF COMBINATION SHAFT, SHOWING POSITION OF ANEMOMETER.

ciency of the upcast shaft, the factors contributing to this are of some importance. I reached the following conclusions:

1. The greater the difference between the temperature of the upcast air and the rock-temperature, the greater the cooling-effect of the shaft.

2. The higher the temperature of the air entering the bottom of the upcast, the greater the cooling-effect of the shaft.

Table XVIII.—*Air-Measurements in Combination Shaft.*

Velocity, Feet per Minute.

Date, 1908.	Pump.						No. 1.					
	1.	2.	3.	4.	5.	Average	6.	7.	8.	9.	10.	Average.
July 16..	110	315	330	200	405	272	312	400	312	350	570	389
Oct. 10. .	180	260	300	370	420	306	140	300	..	..	.	188
Nov. 14 .	560	500	452	500	510	504	630	520	. ...	622	270	510
Nov. 15 .	.....	. . .	... .	.....		353	930	481	... .	630	363	601
Nov. 21 ..	436	453	459	445	490	456	612	488	... .	622	402	525
Dec 27 ..	206	304	320	320	222	274	518	438	... .	526	286	442

Date, 1908	No. 2.						No. 3.					
	11.	12.	13	14.	15.	Average	16.	17.	18.	19.	20.	Average.
July 16....	520	465	560	340	380	453	450	510	560	330	275	425
Oct. 10. .	..	. .	... .	. .	. ..	411	350	370	600	410	422	430
Nov. 14 ..	510	515	.....	530	418	494	520	590	620	406	615	519
Nov. 15..	540	470	... .	310	400	430	461	600	600	390	560	523
Nov 21 ..	...	.	.. .	.. .	.	556	479	617	638	490	584	561
Dec. 27....	548	374	465	367	272	345	461	429	485	312	403	418

Cubic Feet of Air per Minute.

Date, 1908	Pump C.	No. 1.	No. 2.	No. 3	Total.	Average Velocity.
July 16. ..	10,472	11,670	13,790	13,750	49,482	385
Oct. 10 .	11,781	14,640	8,220	12,900	47,551	434
Nov. 11 .	19,404	12,240	9,880	16,470	57,994	514
Nov. 15 ..	13,590	14,424	8,600	15,690	52,309	476
Nov. 21 .	17,556	12,600	11,120	16,830	58,111	524
Dec. 27. ..	10,549	10,608	10,350	12,540	44,047	370
Average..	13,892	12,697	10,293	14,696	51,582	450

Date, 1908.	Time.	Temperature, Shaft-Collar.	Relative Humidity, Shaft.	Surface-Temperature.	Relative Humidity, Surface.	Barometer.
		Deg. F.	Per Cent.		Per Cent.	In.
July 16....	10.00 a.m.	80.6?	100+	65.3	29.9	24.35
Oct. 10... .	....	......	100+	......	... ...	....
Nov 14 ..	3.30 p.m.	88	100+	50.0	...	23.92
Nov. 15 .	3 to 4 p.m.	89	100+	52.9	33.8	23.90
Nov. 21...	3.00 p.m.	89	100+	42.8	65.0	23.70
Dec 27.. ..	1.30 a.m.	84.2	100+	46.0	54.0	24.00
Nov. 16 ..	2.00 p.m.	87.0	100+	61.0	......	24.00
Nov. 17...	3.00 p.m.	88.0	100+	64.0	. ....	24.00
Nov. 18 ..	3.00 p.m.	88.0	100+	55.0	......	23.80

Table XIX.—*Relation Between Depth of Shaft and Cooling of Air-Current.*

Shaft.	Temperature at Foot.	Temperature at Collar.	Depth of Shaft.	Loss in Temperature.	Loss in Temperature, per 1,000 Ft. Depth.	Loss in Temperature, per 1,000 Cu. Ft. Air.	Loss in Temperature, per 1,000 Sq. Ft. Rubbing-Surface Per Min.
	Deg. F.	Deg. F.	Ft.	Deg. F.	Deg. F.	Deg. F.	Deg. F.
Ophir	102.1	88.9	1,600	13.2	8.25	0.22	0.06
Belcher.	81.0	73.9	1,200	7.1	6.00	0.22	0.07
Combination.	95.1	87.6	1,600	7.5	4.70	0.145	0.02

Average temperature of upcast air, $\begin{cases} \text{Ophir, } 95.5^\circ. \\ \text{Belcher, } 77.4^\circ. \\ \text{Combination, } 91.35^\circ. \end{cases}$

3. The lower the velocity, and consequently the smaller the volume of air in the upcast shaft, with given area and rubbing-surface, the greater the cooling-effect of the shaft.

4. The greater the proportion of rubbing-surface to shaft-area, the greater the cooling-effect.

5. Where incast and upcast air are conducted through the same shaft, the cooling-effect of the incast upon the upcast air is very marked.

Regarding the effect of humidity upon the cooling-effect of the shaft, no definite conclusion was reached.

The recording-anemometer used in the Sutro tunnel was placed in the Ophir shaft, but no very satisfactory results were obtained on account of the failure of the instrument to operate in a vertical position. A special fan to operate in a horizontal position was constructed and connected with the recording part of the instrument by means of bevel-gears. The instrument was placed in the pump-compartment of the Combination shaft, calibrated in position by means of small anemometers, and three records taken. The results are given in Tables XX., XXI., and XXII. The factor 0.84 was used in reducing the velocities as given by the instrument to average velocities for the shaft. This factor was determined by taking the average values for the ratio of average velocity and center velocity.

The results from Table XX. are plotted in Fig. 11. The curves show the same characteristics as those of the Sutro tunnel, with the difference that the irregularities are more accentuated. The "pumping-action" of the shaft, due to constrictions, is clearly shown. The shaft is of uniform section, but the air-way leading to the bottom narrows down to about two-thirds the area of both branches of the Sutro tunnel on either side of the Combination partition.

Comparing Tables XVIII. and XXII., the average quantity of air passing is greater by 9,513 cu. ft. per min. for the 24-hr. measurements than for the average of the single measurements. The average minimum 24-hr. measurement compares closely with the average of the results shown by the single measurements. This result is to be expected, for the reason that the afternoon marks the period of the minimum velocities, and most of the single measurements were made in the afternoon.

TABLE XX.—*Air-Velocity Measurements, Combination Shaft.*

Observed Velocities, Miles per Hour.

Time.	Nov. 14 to 15.	Nov. 16 to 17.	Nov 17 to 18.
4 to 5 p.m.	6.20	5.35	5.60
5 to 6 p.m.	5.50	5.50	5.35
6 to 7 p.m.	6.70	6.10	5.60
7 to 8 p.m.	6.70	6.30	5.80
8 to 9 p.m.	6.30	6.30	5.80
9 to 10 p.m.	6.70	6.30	5.95
10 to 11 p.m.	6.30	6.30	6.10
11 to 12 p.m	5.80	6.30	5.60
12 to 1 a.m.	6.30	61.0	6.70
1 to 2 a.m.	6.70	6.70	6.30
2 to 3 a.m.	6.70	6.30	6.20
3 to 4 a.m.	6 80	......	6.30
4 to 5 a.m.	6.70	5.80	6.70
5 to 6 a.m.	......	6.70	5.80
6 to 7 a.m.	......	6.70	6.80
7 to 8 a.m.	6.30	6.10	6.10
8 to 9 a.m.	6.10	5.80	5.60
9 to 10 a.m.	5.80	5.80	5.80
10 to 11 a.m.	5.80	5.35	5.80
11 to 12 a.m.	5.80	5.80	5.35
12 to 1 p.m	5.60	5.35	5.80
1 to 2 p.m.	4.80	5.35	5.35
2 to 3 p.m.	......	5.35	5.10
3 to 4 p.m	......	6.10	5.50

TABLE XXI.—*Average Velocities Over Shaft Section.*

Date, 1908.	Average Velocity.		Maximum Velocity.		Minimum Velocity.		Time of Maximum Velocity.	Time of Minimum Velocity.
	Miles per Hour.	Feet per Minute.	Miles per Hour.	Feet per Minute.	Miles per Hour.	Feet per Minute.		
Nov. 15 to 16...	5.67	547.1	5.71	549.7	4.03	388.8	3 to 4 a.m.	1 to 2 p.m.
Nov. 16 to 17...	5.03	485.3	5.53	533.6	4.49	433.3	1 to 2 a.m.	12 to 3 p.m.
Nov. 17 to 18...	4.87	469.9	5.71	549.7	4.28	413.9	6 to 7 a.m.	2 to 3 p.m.
Average..........	5.19	500.7	5.65	544.3	4.26	412.0	1 to 7 a.m.	1 to 3 p.m.

TABLE XXII.—*Volume of Air in Cubic Feet per Minute.*

Date, 1908.	Average.	Maximum.	Minimum.	Shaft-Temperature.	Shaft-Humidity.	Surface-Temperature.	Barometer.	Time.
				Deg. F.	Per Cent.	Deg F.	In.	
Nov. 15 to 16......	66,745	67,063	47,441	89	100+	52.9	23.9	Nov. 15, 3 p.m.
Nov. 16 to 17.....	59,210	65,098	52,734	87	100+	61.0	24.0	Nov. 16, 2 p.m.
Nov. 17 to 18......	57,330	67,063	50,498	88	100+	64	24.0	Nov. 17, 3 p.m.
				88	100+	55	23.8	Nov. 18, 3 p.m.
Average............	61,095	66,408	50,224	88	100+	58.5	23.9	

10. *Supplementary Ventilation.*—At the Ward shaft a No. 200 American, three-quarter housing, vertical-discharge, suction fan, driven by a 50-h.p. induction-motor, is used to ventilate the workings of the shaft; and, as sinking is the only work, with the exception of the pump-stations, practically the full capacity of the fan is used for this work. The incast air passes down the two hoisting-compartments, and is deflected in part by a canvas curtain at the 2,475-ft. station (pump-station). The curtain is one-half the width of the compartment, and is suspended by its upper edge from a wall-plate on a level with the roof of the station. The lower edge hangs loose over the guard-bar in front of the compartment. This gives a curved surface

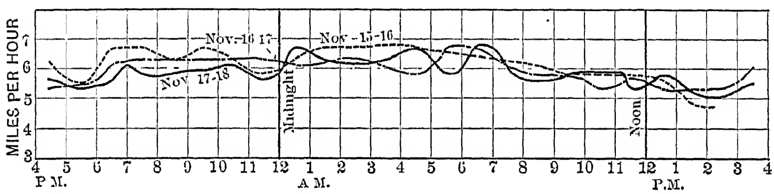

FIG. 11.—CURVES SHOWING RELATION BETWEEN OBSERVED VELOCITIES AND TIME OF DAY, COMBINATION SHAFT.

and deflects enough air to ventilate the station, while allowing the cage, in passing down, to sweep it out of the way. The greater part of the air passes down to the sump and then rises through the pump-compartment, which is bratticed off from the hoisting-compartments.

The discharge of the fan was measured several times. Five points were taken in the discharge section, and the average of the anemometer-readings was taken as the discharge velocity. Table XXIII. gives the results.

On Dec. 13, at the 2,475-ft. level, the temperature in the station close to the down-cast compartments was 91.4° F., and the relative humidity 93.6 per cent.; in the upcast compartment on the same level the temperature was 104° and the relative humidity 100 per cent. plus. The upcast air was heavily charged with vapor.

The ventilating-effect of the fan is supplemented by the chimney-effect of the shaft. Calculating this effect, and not allowing for leakage through the brattice, a suction of 1.93 in. of water was obtained. This would leave 2.07 in. as the effect of the

TABLE XXIII.—*Air-Measurements of the Ward-Shaft Fan.*

Date, 1908.	1.	2	3.	4.	5.	Mean Velocity.	Cu. Ft Per Min
July 19.....	2,842	4,200	3,556	2,842	3,576	3,476	105,280
Aug. 8......	2,248	3,860	3,464	2,280	3,292	3,028	91,597
Nov. 22....	2,760	3,428	3,044	2,628	2,944	2,959	89,510
Dec. 13.....	2,360	3,552	3,272	2,552	3,272	3,002	90,810
Mean	2,456	3,613	3,260	2,486	3,169	2,996	90,639

NOTE.—Only the last three readings were averaged, as the anemometer was not calibrated for the first reading.

Date.	Fan Discharge.		Air.		Time.
	Temperature.	Relative Humidity.	Temperature.	Relative Humidity.	
	Deg. F.	Per Cent.	Deg. F.	Per Cent	
July 19......	......	......	......	......	9.00 a.m.
Aug. 8.......	90.0	37.0	88.34	14.0	9.00 a.m.
Nov. 22......	59.0	90.4	36.7	74.9	7.30 a.m.
Dec. 13......	55.4	94.3	36.5	56.5	12.50 a.m.

Water-gauge on suction side of fan gave 4 in. of water.

fan. The catalogue-rating of the fan is given as 75,700 cu. ft. per min. at 189 rev. per min. and an expenditure of 53.3 h-p. This would give, by difference, 14,939 cu. ft. per min. as due to chimney-draft. Calculation shows the chimney-effect to be equivalent to 17,088 cu. ft. per min. The difference between these two results is undoubtedly due to the catalogue-rating of the fan, which can be taken only in a very general way.

At the Ophir shaft a B. F. Sturtevant No. 16, double width, multi-vane fan is being installed. This fan is rated to deliver 140,000 cu. ft. per min. against a maintained suction of 2 in. of water at a speed of 200 rev. per min. with an expenditure of 100 horse-power.

The plan of the fan-arrangements at the Ward shaft is shown by Fig. 12 and at the Ophir by Fig. 13.

The distribution of the fans underground is given in Figs. 4 and 5, and the details of the fans in Table XXIV. The total horse-power of all motors used for ventilating purposes is 315; of this 150 is used on the surface and 165 underground. The actual horse-power in use, excluding the Ophir fan, which is being installed, is approximately 163.2. Of this 54 is used on

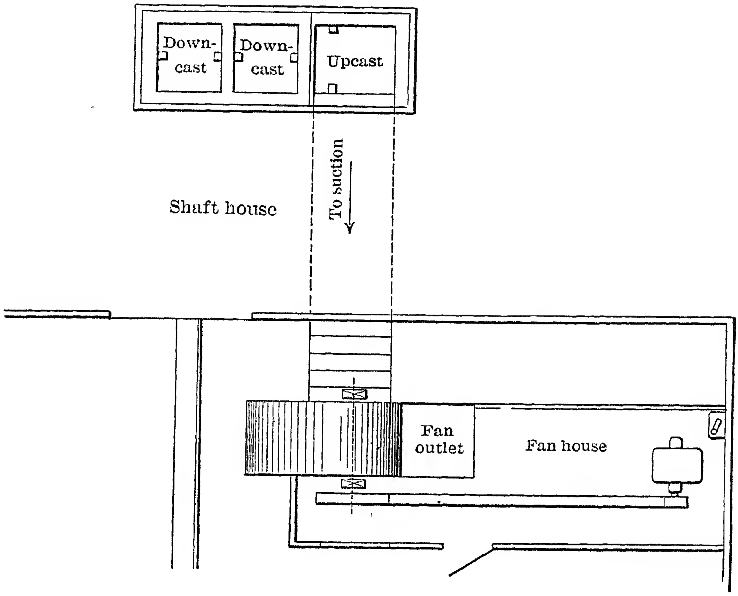

FIG. 12.—PLAN OF FAN AT WARD SHAFT.

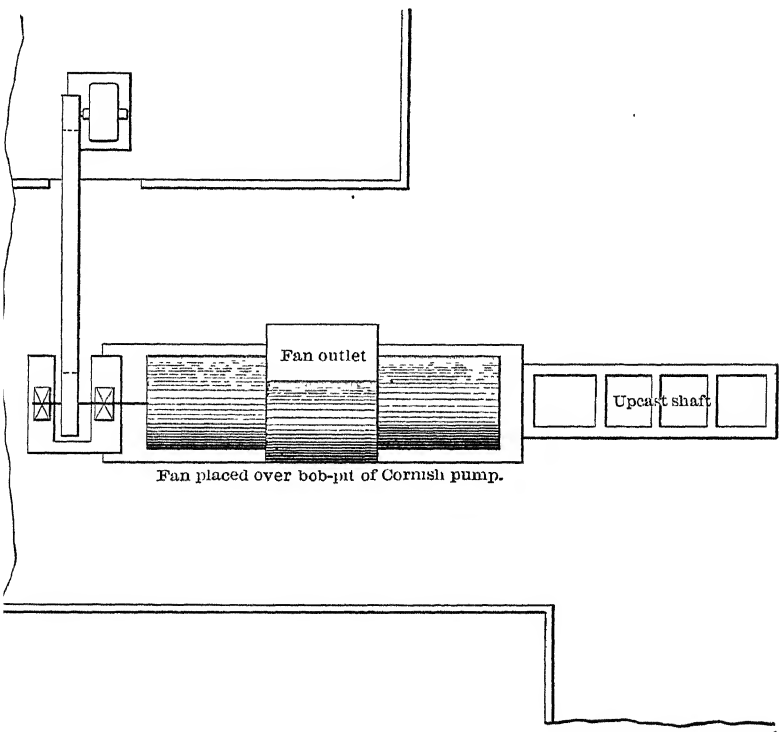

FIG. 13.—PLAN OF FAN AT OPHIR UPCAST.

TABLE XXIV.—*Fans in Use on the Comstock.*

Position.	Fan.	Motor.	Horse-Power.	Speed.	Air.	Diameter of Inlet.	Area of Inlet	Diameter of Outlet.	Area of Outlet	Diameter of Fan-Wheel.	Width of Fan-Wheel.	Notes.
		H.-P.		Rev. per Min.	Cu. Ft. Per Min	In.		In.				
Ward shaft	No. 200 metal exhauster	50	......	200	90.639	76	31.5 sq. ft.	....	31.25 sq. ft	10 ft.	47 in.	
Ophir shaft	No. 16 double mine-fan	100	....	200	140,000	70	53.4 sq. ft	....	53.8 sq. ft.	6 ft. 8 in	7 ft 4 in.	
Hale and Norcross, Sutro tunnel	15-in. metal exhauster.	10	6	1,400	3,761	15	176 sq in.	15	162 sq in	22 in	12.75 in	
Savage mine, Sutro tunnel.	15-in. metal exhauster	10	6	1,000	2,357	15	176 sq in.	15	162 sq in	22 in	12.75 in.	
Gould and Curry, Sutro tunnel	15-in. metal exhauster	10	6	1,000	....	15	176 sq in.	15	162 sq in.	22 in	12.75 in	Not in operation.
Ward pump-station, 2,475 level.	15-in metal exhauster.	5	2	....	....	15	176 sq. in	15	162 sq. in	22 in	12.75 in	
Union, 2,000 level	15-in. metal exhauster	10	8	1,680	4,287	15	176 sq in.	15	162 sq in	22 in	12.75 in	Suction and exhaust
Ophir, No. 1 fan	15-in. metal exhauster	15	4	1,000	3,791	15	176 sq in	15	162 sq in	22 in	12.75 in	
Ophir, No. 2 fan	6-ft. wooden fan	20	24	515	11,566	29	1 009 sq in	20 by 20	400 sq. in	72 in	20 in	
Ophir, No. 3 fan	4-ft. wooden fan.	5	4	360	3,323	20	406 sq in	16 by 16	256 sq in.	52 in	16 in	
Ophir, No. 4 fan	6-ft. wooden fan	15	9.6	340	....	29	1,009 sq in	20 by 20	400 sq in	72 in	20 in	
Ophir, No. 5 fan	6-ft. wooden fan	20	16	500	....	29	1,009 sq in.	20 by 20	400 sq in	72 in	20 in.	
Ophir, No. 6 fan	15-in. metal exhauster	20	14.4	1,500	5,389	15	176 sq in.	15	162 sq in	22 in	12.75 in	
Ophir, No. 7 fan	15-in. metal exhauster	....	....	....	....	15	176 sq in.	15	162 sq in	22 in	12.75 in	Not in operation.
2,150 Ophir, north workings	4-ft. wooden fan	5	1.2	360	....	20	406 sq in.	16 by 16	256 sq in	52 in	15 in	
1,000 level, Yellow Jacket	15-in. metal exhauster.	10	....	1,000	....	15	176 sq. in	15	162 sq. in	22 in	12.75 in.	Not in operation.
Overman-Caledonia	15-in. metal exhauster	10	....	1,680	4,960	16	201 sq. in	16	201 sq in			

TABLE XXV.—*Accession of Moisture, Incoming Air-Currents.*

Nov. 22, 1908.	Temperature.	Absolute Humidity.	Increase Humidity.	Distance.	Exposed Surface.	Volume.	Velocity.	Area.	Perimeter.	Area / Perimeter.	Grams per Square Foot per Hour.	Grams per Square Meter per Hour.
	Deg. F.	Gr. per Cu. Ft.	Gr. per Cu. Ft.	Ft.	Sq. Ft.	Cu. Ft. per Min.	Ft per Min.	Sq Ft.	Ft.			
Sutro Tunnel.												
Portal	46	2 08										
0–5,200	71.6	6.21	4.13	5,200	182,000	32,181	450	71 4	35	2.01	43 8	30 553
5,200–6,800	73	7.29	1.08	1,600	56,000	32,181	450	71.4	35	2 04	37 2	25.949
6,800–8,000	85.2	11.79	4.50	1,200	42,000	32,181	450	71.4	35	2.04	206.9	144.326
8,000–10,000	91.9	15.23	3.44	2,000	70,000	32,181	450	71 4	35	2 04	94 9	66.200
10,000–13,000	95	16 05	0.82	3,000	105,000	32,181	450	71.4	35	2.04	15 1	10 533
13,000–17,000	95	16.64	0 59	4,000	140,000	32,181	450	71.4	35	2.04	8 1	5 650
17,000–18,000	94.4	16.99	0.35	1,000	35.000	32,181	450	71 4	35	2 04	19.3	13 463
18,000–18,700	91.5	17.09	0.10	700	24,500	32.181	450	71.4	35	2 04	7 9	8.734
0–18,700	94.5	17.09	15.01	18,700	654,500	32,181	450	71.4	35	2.01	44 0	30 693
Ward S. connection, Dec. 13, 1908	87.8	11 147				South lateral.						
50 feet S. of H. & N. fan	108.86	14 184	3.037	2,720	89,760	10,143	161	63	33	1.90	20 59	14.363
North lateral partition	79.2	4.918				North lateral.						
C. & C. shaft connection	93 2	15.429	10.481	921	30,393	3,378	54	63	33	1.90	69.89	48 755
Drifts.												
2,350 west drift, section K	84.2	11.00										
2,350 west drift, near winze	98.6	18.77	7.765	530	12,190	11,667	413	28 25	23	1 23	561 4	391.62
2,350 east drift, end of pipe	89.6	5.93										
2,350 east drift, turn of drift, C.C	97.3	7 74	1.806	165	3,877	1,996	63	31 5	23 5	1.34	55.8	38.923
2,150 N. drift, section A	87.4	4.68										
No. 6 fan	98.6	7.123	1.443	660	15,510	7,024	223	31.5	23.5	1.34	39.2	27 345
Shafts.												
C. & C. surface	38.5	0.868										
C. & C., 2,150	59.7	3.944	3 076	1,900	82,650	33,212	431	77	43.5	1.77	74.17	51 739
Yellow Jacket, surface	34.7	1.036										
Yellow Jacket, 900 level	49.6	3.760	2.73	900	33,750	10,665	166	64 2	37 5	1 71	51 76	35.106
Union S., surface	46.8	0 766										
Union S., 2,000 level	65.1	5.688	4 922	2,000	134,000	13,814	109 6	126	67	1 88	30.1	21.206

the surface and 109.2 underground. With the Ophir fan in use the total horse-power approximates 263.2.

11. *Air-Pipes.*—The extensive use of air-pipes warrants a detailed description.

The air-pipe is in 12-ft. lengths, 11-in. and 15-in. diameter being commonly used. No. 20 galvanized iron, in sheets 36 in. wide, cut to the necessary lengths for the diameters required, and with longitudinal seams punched, is delivered to the mine shop and there made into pipe as required. Longitudinal seams are riveted with $\frac{3}{16}$-in. rivets, 2.5-in. pitch; girth seams, 4-in. pitch. Longitudinal and girth seams are soldered. Joints are made with a bell, which is riveted and soldered to the pipe. Laps are 0.75 to 1 in. The mouth of the bell is strengthened with $\frac{3}{16}$-in. wire. The diameter of the mouth is 16 in. and the depth 4 in., not including the lap. This gives an over-all length of 12 ft. for each section.

The bell-joint is known as the "Southwell" joint. Elbows are made with an 18-in. radius, measured to the center of the pipe, and are provided with a bell-joint; the inside diameter of the mouth is 15.5 in.; the outside, 16.25 in. Rivets on transverse joints are pitched 2.5 in., and on longitudinal joints 1.5 in. Six pieces, including the bell, are required for each joint. The opposite end of the elbow has a diameter of 15.75 in. Right-angle tees, used for branches, are provided with two 11-in. dampers, having a clearance of $\frac{3}{16}$ in., and 15-in. dampers with a clearance of from $\frac{1}{8}$ to $\frac{1}{16}$ in. The dampers are riveted to a flattened $\frac{5}{8}$-in. iron rod, one end of which is bent so as to form a 6-in. handle. The damper is cut in the blacksmith-shop from $\frac{1}{8}$-in. sheet iron. Elbows of 45° are made from three pieces, including the bell. "Y" pieces and special breechings for the fans are made as required; 20-in. pipes, occasionally used, are made from No. 18 or No. 20 galvanized sheet-iron, with longitudinal seams riveted on 2.5-in. pitch, and girth seams from 5 to 6.75-in. pitch. The inside diameter of the bell-mouth is 20.75 in.; the outside, 21.25 in.; and the depth of the bell is 5 inches.

Air-pipes are suspended by U-shaped iron straps, made from $\frac{3}{16}$- and $\frac{1}{4}$-in. round iron, each end of the U being provided with a 3-in. point. The straps are driven into the timbers, three straps to each 12-ft. section. Where the air-pipe is placed

in untimbered drifts, it is suspended by ropes or wire from plugs driven into drilled holes, or else from sprags wedged into place, and into which the iron strap-points are driven.

The bell-joints are made tight by wrapping several times with a tarred canvas strip 7 in. wide. Six or seven wrappings of 0.25-in. tarred cord are used to hold the canvas in place. New pipe, carefully wrapped in the manner described, allows very little air to leak through at the joints. Data on leakage will be given later in this paper.

If protected from acid mine-waters, and not battered by blasting, the life of an air-pipe is indefinite; but in wet shafts or where exposed to acid waters they frequently have to be replaced.

The shop-cost of 15-in. air-pipe is $8.50 per section; the cost of installation is nominal. The placing of a 15-in. air-pipe approximates $9.25 per section, or $0.77 per linear foot.

Air-Pipe Leakage.—Leakage depends very largely upon the care taken in wrapping the joints. If the ends are much battered the leakage may be considerable. Pressure and velocity of the air are minor factors.

A well-wrapped air-pipe extends from the Mint shaft to the Hale and Norcross workings. Connection is made with a fan at a distance of 1,500 ft. from the shaft. The discharge from this fan, at a distance of 230 ft., showed a loss of 411 cu. ft. per min., or a delivery of 89 per cent. of the inflowing air. The loss per joint is 2.93 cu. ft. per min. A water-gauge at the Mint shaft showed a pressure into the air-pipe of $\frac{13}{16}$ in. The continuation of the pipe, 267 ft. of 15-in. and 268 ft. of 11-in., or 535 ft. in all, gave an additional leakage of 1,941 cu. ft. per min. The pipe was not well wrapped, and the loss per section figures out 44.1 cu. ft. per min. In this case the suction created by the fan undoubtedly reduced the leakage on the 1,500 ft. section.

In the north lateral of the Sutro tunnel 120 ft. of 20-in. air-pipe showed a leakage of 122 cu. ft. per min., or 12.2 cu. ft. per section. The pipe was not wrapped, but was very carefully placed together.

In the Gould and Curry mine a leakage in 825 ft. of 15-in. and 11-in. pipe measured 437 cu. ft. per min., or 6.4 cu. ft. per min. per section. The pipe was partly wrapped and the air-pressure measured $\frac{13}{16}$ in. The fan was not in use.

In the Savage workings, with fan in operation and 15-in air pipe, the leakage amounted to 1,610 cu. ft. per min. The length of 555 ft. of 15-in. pipe beyond the fan and 180 ft. of 11-in. pipe gives a leakage of 26.4 cu. ft. per min. per section. In this case most of the leakage took place on the last 180 ft. of 11-in. pipe.

Dampers showed a variable leakage, depending upon the perfection of the fitting and the care used in closing. Measurements showed from 10 to 180 cu. ft. per minute.

Two canvas curtains were measured: one showed a leakage of 1,635 cu. ft. per min. over an area of 35 sq. ft.; the other, 3,980 cu. ft. per min. over an area of 20 sq. feet.

The general conclusion is that leakage can be controlled within narrow limits by carefully wrapping the joints. A mile of 15-in. pipe with a leakage of 2.93 cu. ft. per min. per section would give a total leakage of 1,289 cu. ft. per min., and with an inflow of 4,000 cu. ft. per min. a delivery of 67 per cent. of the air is possible. This would be more than sufficient to ventilate an ordinary working-face. It is not always desirable to have a pipe tightly wrapped up to the face. Mr. Higginson informed me that with pipe tightly wrapped the blasts, if heavy, will cause much loss by collapsing. The practice is to wrap the pipe up to within from 200 to 300 ft. of the face and then leave the remainder unwrapped. Old and battered pipe is used close to the face.

Stops are constructed from timber or canvas; 1-in. rough boards, battened, are commonly used; doors are made from the same material; clay is used to stop small cracks and openings.

V. Measurement of Ventilating-Currents.

1. *Temperature.*—The mine-measurements were made with an ordinary Fahrenheit thermometer of good grade. The instrument was suspended from a timber, and in approximately the center of the air-way. Unfortunately, this thermometer was not available for calibration. My observations were made with chemical thermometers carefully matched and compared.

2. *Velocity.*—All upcast shafts are housed in by wooden chimneys, which conduct the air outside the shaft-houses. Access to a shaft is had through doors, and from them most of the section may be reached. Five measurements were made in

as many positions in each compartment. The center and four corners were the positions selected, as shown by the numbered circles in Fig. 10. The corner position was taken 1 ft. from the timbers each way. The anemometer was held stationary in each position, and the velocity determined over a time-interval varying from 0.5 to 2 min. Observed velocities were corrected by the calibration-curve furnished by the maker of the instrument. Measurements were not taken by moving the instrument over the whole section on account of the difficulty of reaching some of the compartments. In part of the time during which measurements were made no work was done in the shafts, but during the remainder shaft-crews were at work in all three upcasts. The housing of the shaft at the collar prevented eddy-currents, and consequently all measurements were made in the plane of the shaft-collar. The time required to make from 15 to 20 separate measurements on one shaft varied from 30 to 50 min. Table XVIII. presents a complete set of observations for the Combination shaft.

The down-cast air was measured in one shaft only and at one time, since in all of the down-cast shafts work was going on. Difficulty was experienced on account of eddy-currents, even though the anemometer was held some distance below the collar of the shaft.

The center velocity of each compartment generally exceeded the corner velocities. The ratio between the two was calculated and an average of 1.19 obtained for the Combination, 1.09 for the Ophir, and 1.19 for the Belcher shaft. The reciprocal of 1.19 is 0.84, and this was used to calculate average velocities from center velocities. As an example of the use of this method for determining the average velocity, the following figures are given: The Ophir shaft was measured by both methods, Dec. 27, 1908. The average of five readings for each compartment was 653, 652, and 675 ft. per min., and the center velocities were 778, 732, and 732. Computing from these figures by the use of the constant 0.84, the following results were obtained: 690, 651, and 651 ft. per min. The average velocity by the 15 readings was 660, and the calculated velocity 664 ft. per minute.

Sections were taken in straight portions of the drifts where-

ever these were available, and timbered sections were used in preference to the untimbered, but no special steps were taken to prepare sections of uniform cross-section. The anemometer was slowly moved over the whole section for a period varying from 0.5 to 2 min. Duplicate readings were generally taken.

On the whole, the measurement of the air in the air-ways of a mine, without special provision for smooth sections, is approximate only, and it is not possible to make very accurate measurements without resorting to extreme measures. My measurements were made during the working-hours of the mines, and without special provision for smooth sections.

The anemometers were of the Biram type, one 4-in. reading to 1,000 ft., one 4-in. reading to 100,000 ft., and one 6-in. reading to 1,000 ft. The recording-anemometer was a Julien P. Friez instrument, of the type used at the government meteorological stations. A sling-psychrometer was used in determining humidities, and the psychrometric tables published by the U. S. Weather Bureau were used in reducing results.

VI. Cost and Operating-Expense.

It is almost impossible to obtain detailed figures on the cost of the ventilating-system, but fairly-accurate estimates may be made. The following estimate was made by taking the cost of motors and fans and adding 20 per cent. for cost of installation. The 15-in. air-pipe in place was estimated as $0.77 per ft. and 11- and 20-in. proportionally:

Cost of supplementary ventilating-plant:	
Motors,	$6,000
Fans,	6,000
Air-pipe,	10,000
	$22,000

Note.—The above does not include the cost of the fan and pipe in the Overman and Caledonia mines, or the cost of electrical conduits.

Depreciation, maintenance, and operating-expense:	
Depreciation and maintenance assumed as 10 per cent. of plant-cost, or $2,200 per annum, or per month,	$183.30
Operating-cost:	
Power: 165 h-p., at $5 per h-p-month,	825.00
1 electrician, at $4 per day,	120.00
Incidentals,	50.00
Total expense per month,	$1,178.30

Note.—Shift-bosses and miners take immediate charge of fans and motors, and no extra men are employed except the electrician.

The expense of maintaining the upcast shafts may be properly charged against the cost of ventilation, and this is a large item, but at present no figures are available. During the past year shaft-crews of from four to six men have been employed on single shifts in all three upcast shafts for periods of from two to six months. On an average this work has to be done every second year.

On the foregoing estimate of expenses per month, the cost ot each 10,000 cu. ft. of air passing through the mines is $0.00116, not including the cost of repair of upcast shafts.

VII. Underground Humidity.

In summer the air entering during the day is of low and in winter of moderate humidity. The temperature is neither excessively low nor high. The greater heat of summer is partly compensated for by the lower humidity, while the higher humidity of the winter months is compensated for by the lower temperature. The temperature of the entering air is of more importance than the humidity. The air-current begins to rise in temperature and absorb moisture as soon as it enters the underground workings. How rapidly it changes depends upon the velocity of the air-current, the difference of temperature between the air-current and the surrounding rocks, the relative humidity, and exposure of the air-current to moisture. The effect of these factors may best be studied from the three following examples:

1. *The Sutro Tunnel.*—Table II. and Fig. 2 are referred to. The temperature of the incast air reaches a maximum at a point 11,000 ft. from the portal. The temperature of the water controls the air-temperature. Humidity begins to increase as soon as the air enters the tunnel. For the first 6,800 ft. the air does not come in contact with the drainage-water, and consequently the moisture must come mainly from the walls.

The amount of moisture from this source is 43.8 grains per square foot of exposed surface per hour. Much of this part of the tunnel is untimbered. At the 6,800-ft. station the air comes in contact with the drainage-water, and absorbs moisture until it is almost saturated at a point 10,000 ft. from the portal, or after moving over the water for a distance of only 3,200 ft. The approximate exposed water-surface is 19,200 sq. ft. (average width

of stream taken as 6 ft.). On the basis of the exposed water-surface the rate of accession of water-vapor is 798.5 grains per square foot per hour. The rapidity with which the air takes up moisture is remarkable. The rates for different parts of the tunnel have been calculated and are to be found in Table XXV. The total exposed surface, both rock walls and water, has been taken in calculating the table in preference to the water-surface

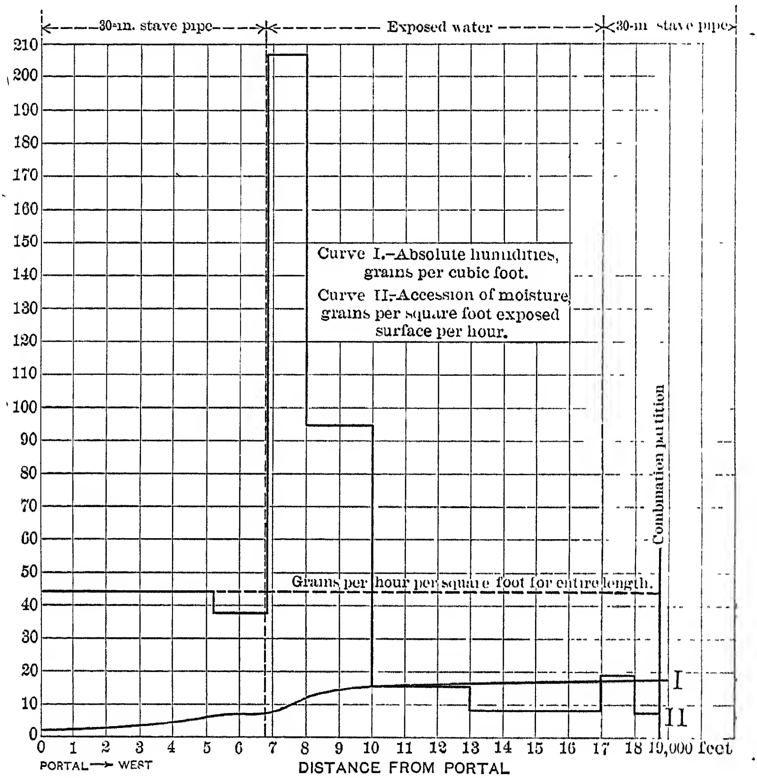

FIG. 14.—CURVES OF ABSOLUTE HUMIDITIES AND RATE OF ACCESSION OF WATER PER HOUR IN THE SUTRO TUNNEL, NOV. 22, 1908.

alone. The results have been also graphically represented in Fig. 14. The rate for the whole tunnel up to the Combination partition is 44 grains per square foot per hour. For purposes of comparison, both north and south laterals for a portion of their length were calculated. The south lateral is timbered for about one-half the distance taken. Very little water drops from the walls, and the drainage-water is carried in a stave pipe, so that

practically all the moisture comes from the walls. A rate of 20.59 grains was found. The north lateral, on the other hand, has open drain-boxes and more or less water is present, although the main flow of drainage-water is carried in stave pipes. The rate is 69.89 grains.

2. *Mine-Drifts.*—Three drifts have been calculated: the very hot west drift of the 2,350-ft. level; the east drift of the same level; and a portion of the 2,150-ft. drift. Although the west drift of the 2,350-ft. level is snow-shedded, the hot vapors find their way through with considerable freedom and give a high rate, 561.4 grains per square foot per hour. The east drift of this level is comparatively dry and practically out of the hot zone. It gives a much lower rate, 55.8 grains. The 2,150-ft. drift is also snow-shedded, but, since the lower workings have drained off the hot water to a great extent, the accession of moisture is very much less, 39.2 grains per hour.

3. *Down-Cast Shafts.*—Three down-cast shafts have been calculated: the C. & C. down to the 2,150-ft. level; the Yellow Jacket to the 900-ft. level; and the Union to the 2,000-ft. level. The range of temperature and the relative humidity at different levels in these shafts are shown in Fig. 15.

The C. & C. shaft is not a very wet shaft, but considerable moisture from the drainage-water is present from the 1,750-ft. to the 2,150-ft. level, which accounts for the high rate of 74.17 grains per square foot per hour. The Yellow Jacket is moderately wet, while the Union is quite dry. The rates are 51.76 and 30.4 grains respectively.

A better idea of the amount of water taken up by an air-current under high-temperature conditions may be gained by considering the total amount of moisture discharged with the air at the upcasts. The total amount in all upcasts is 519,788 lb. per 24 hr., divided as follows: Combination, 146,330; Ophir, 172,140; Belcher, 58,941; Ward, 142,377 lb. These quantities have been calculated on the average amounts of air as given in Table III. Assuming an average temperature of 60° F. and a relative humidity of 50 per cent., the down-cast air brings into the underground workings 127,990 lb. of moisture per 24 hr., which leaves 391,798 lb. of water per 24 hr. as the amount taken up by the air-currents. The calculated result is somewhat less than the actual amount.

The unit used in comparing different mine-workings is the rate of accession of water-vapor per square foot of exposed surface per hour, and is calculated from the following ratio:

$$\frac{\text{Volume of air in cubic feet per minute} \times \text{increase in absolute humidity per cubic foot for the length under consideration} \times 60 \text{ min.}}{\text{Square feet of exposed surface in length under consideration, or length of air-way} \times \text{perimeter.}}$$

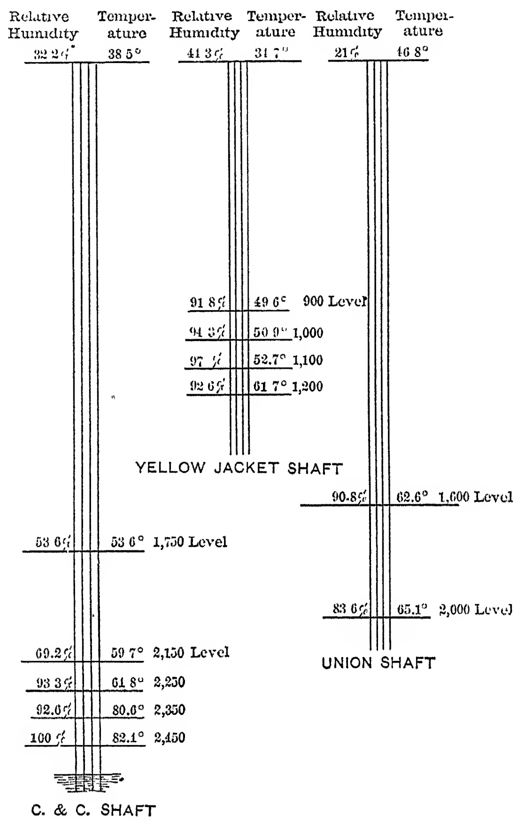

FIG. 15.—RANGE OF TEMPERATURE AND RELATIVE HUMIDITY IN DOWN-CAST SHAFTS.

VIII. CONTROL OF HUMIDITY AND HEAT.

Control of humidity and heat is a matter of considerable importance as affecting the efficiency of the miners. High temperatures and saturated air are conducive to low labor-efficiency, while, on the other hand, high temperature with low humidity does not lower efficiency to an unreasonable extent. The ex-

amples in the previous section show that if water can be kept away from the air a moderate humidity is possible, but this cannot always be done. Careful boxing of drains and boarding-off of hot water are beneficial; the carrying of large quantities of drainage-water in closed conduits, such as the stave pipe, is very effective, but, on the whole, the mine-drift or tunnel cannot be considered an ideal air-way under high-temperature conditions. However carefully the drainage-water may be carried, the emanation of water-vapor from the walls and surfaces of the timbers still remains as an important humidifying-agent. The construction of moisture-tight walls for the drifts is out of the question, and the use of excessive volumes of air is impracticable. The carrying of air through sheet-metal conduits of moderate cross-section is the only practicable way of delivering air-currents of low humidity to working-places where a high temperature prevails, and this method is in general use upon the Comstock. The small size of the air-pipe as compared with the drift requires the use of power-driven fans to force a sufficient quantity of air. The working-face of a dead end, such as a drift, is perhaps the most difficult to maintain in proper condition, and as a consequence a number of measurements were made in the ends of drifts ventilated by pipes. These measurements are given in Table XXVI.

With the exception of two cases the air is delivered practically either at the temperature of the working-face or a little above. The two exceptions indicate that if a sufficient volume is forced through the pipe the air can be delivered lower in temperature than the working-face, although usually the volume is such that even with short distances the air is delivered at the same temperature as the drift through which the return current is passing. The quantity of air varies from 500 to 1,200 cu. ft. per min., in ordinary cases, up to from 2,000 to 4,000 cu. ft. per min. in cases in which an excessive temperature is to be overcome. The humidity of the discharged air depends upon the humidity of the air entering the pipe or fan and the temperature of the discharged air, and for this reason the air should be drawn from a current low in both temperature and humidity. The relative humidity of the discharged air ranged from 20 to 41 per cent., while the relative humidity at the face, usually 20 to 25 ft. beyond, ranged from 22 to 74.4 per cent.

TABLE XXVI.—*Air- and Temperature-Measurements at Drift-Ends.*

Place.	Temperature of Discharged Air.	Temperature of Working-Face.	Relative Humidity of Discharged Air	Relative Humidity of Working-Place.	Distance of Working-Face from Pipe-End	Cubic Feet of Air per Minute	Velocity of Discharge	Diameter of Air-Pipe.
	Deg. F.	Deg. F.	Per Cent.	Per Cent	Ft.		Ft. per Min.	In
Large Volumes of Air.								
2,350 C. & C. shaft, west drift	82.4	110	....	100	25	4,160	3,492	15
2,350 C. & C. shaft, east drift	89.6	96.8	40.6	50.7	40	1,996	2,050	15
Moderate Volumes of Air.								
Gould and Curry, Sutro tunnel	91.0	91	31	31	12	989	1,521	11
Savage, 75 ft. above Sutro tunnel	92.6	92.1	37.3	51	30	894	1,356	11
Chollar, Sutro tunnel	104.5	....	23.8	....	....	934	1,416	11
Chollar, Sutro tunnel	106.2	106.2	22	22	10	721	1,104	11
Hale and Norcross, cross-cut	106.1	107.6	21.8	32.2	15	941	1,426	11
Hale and Norcross, cross-cut	109.4	109.6	20.0	29.8	25	688	1,054	11
2,000 level Union, NW. cross-cut	99.1	93.6	38.3	43.4	20	1,271	1,042	15
2,000 level Union, south drift	98.6	....	40.8	....	30	971	796	15
2,350 C. & C., north workings	104.5	....	36.5	....	....	544	490	15
1,200 level, Caledonia	86.0	87.8	41.0	74.4	25	792	1,200	11
No Active Circulation of Air.								
2,000 level, Union, east cross-cut		100.9	....	74.8				
1,100 level, Yellow Jacket, north drift to Imperial		89.9	....	86				

The first drift in the table, the west drift of the 2,350-ft. level of the C. & C., illustrates abnormal conditions. Great heat and a large quantity of moisture emanated from the rock walls, which had not at the time been closed off by boarding, and this charged the air-current to saturation and raised the temperature 27.6° F. in a very short distance. The east cross-cut of the Union on the 2,000-ft. level is a dead end, in which practically no air was circulating, and which was also dry. The north drift on the 1,100-ft. level of the Yellow Jacket is also a dead end, without circulating air, but with a small amount of water on the floor of the drift. In neither of these drifts was the air saturated with moisture, but in both it was quite oppressive.

The principle and use of the air-pipe as a means of delivering low-humidity air-currents is of course no new thing. Church comments upon the efficiency of the system, and in particular notes that by forcing the air through metal pipes the relative humidity of the air-current is lowered and it becomes quite effective in absorbing the excessive perspiration of the miners. The only improvements to be noted in Comstock practice since the time of Church's observations, about 30 years ago, are an increased velocity and a larger quantity of air, larger air-pipes, the use of high-speed metal fans instead of the slower wooden fan, the use of electric motors instead of compressed-air engines, and the use of more power.

Supersaturation of air-currents, resulting in the production of fog, is encountered in all of the upcast shafts near the surface, and in all places underground where cool currents mix with hot saturated air. The moisture condenses on timbers and rock-surfaces, and frequently causes the rock to swell, disintegrate, and cave. By stimulating the growth of fungi it also indirectly causes the timbers to rot. In only one place is any attempt made to remedy such a condition, and in this case a small amount of cool air is introduced into the air passing up the Ophir incline. Trouble was experienced in this incline by the condensed moisture causing the "back" to swell, and the remedy was found effective. It is evident that the introduction of cool air lowers the temperature of the upcast air and thus reduces the efficiency of the upcast, and if too much air is introduced the opposite from the desired effect would be obtained. Passing the air through the shortest and driest

air-ways to the upcast, and the careful stopping of all leaks of cool air, would be a more effective way than that now followed.

Some observations upon the cooling-effect of large volumes of air were made. The most striking results were in the case of the hot west drift on the 2,350-ft. level of the C. & C. Before the connection was made the maximum temperature of this drift varied from 126° to 130° F., and this in spite of forcing 4,160 cu. ft. of air per minute into the face. The air in the drift was saturated and would not support a candle-flame. After making the connection and before boarding-off the walls of the drift, the temperature dropped to 110°, with a volume of air passing of 10,350 cu. ft. per min. The air-current was supersaturated. After snow-shedding, the temperature dropped to 98.6°, with 11,667 cu. ft. per min. passing and a relative humidity of 98.8 per cent. In the Hale and Norcross cross-cut, on Dec. 28, the rock-temperature measured 113° and the air-temperature 109.6°. The reduction in temperature of more than 3.4° was effected by an air-current of 688 cu. ft. per minute.

The most effective means for the control of an excessive temperature is the tight boarding of water-boxes carrying hot water, the use of boards and battens for the sides and tops of drifts, and passing large volumes of air as cool as possible.

IX. Physiological Effect of Working in High Temperature and Humidity.

Much has been written regarding the effects upon the human system of poisonous and other gases, but very little is to be found in mining-literature concerning high temperature and humidity. Le Neve Foster says: [5]

"In still and saturated air it is hardly possible for men to do hard continuous work above 80° or 85° F., even when stripped to the waist. At higher temperatures in saturated air the amount of work possible becomes less and less, and the body temperature may rise rapidly, though men accustomed to the heat can bear it much better than others. At temperatures above about 90° by the wet bulb it is only possible to work for short periods, and it becomes difficult even to remain without working. Thus, at a temperature of 93° in still and saturated air, I found that, though I was stripped to the waist and doing practically no work, my temperature rose 5° in two hours, and was still rising rapidly when I found it necessary to come out. On the other hand, it is a well-known fact that if the air is dry much higher temperatures can be borne with ease and comfort. In collieries

[5] *Investigation of Mine Air*, Sir C. Foster and J. S. Haldane, p. 151 (1905).

where the air is fairly dry and in motion, men can work well at a dry-bulb temperature of 90°, or even 100°, and in hot climates with very dry air much higher temperatures are not oppressive."

Eliot Lord[6] discusses the permanent effect of high heat upon the system, and reaches the following conclusion:

"The ultimate effect of this extreme heat on the miners' constitution is not so easily noted. The mine levels differ so materially in temperature, and the assigned station of a miner is so frequently changed from one cause and another, that it is impossible to obtain at present complete comparative data. That prolonged labor in a hot, impure atmosphere will assuredly shorten life appears indisputable; but whether the system is permanently or materially injured by intermittent working under those conditions is more questionable. The power of recuperation appears extraordinary, and, unless the strain is intense and frequent, no lasting injury may be inflicted. The limits of permissible strain will, of course, vary with the relative power of endurance. The action of all the bodily organs appears to be stimulated by the heat, with the exception of the stomach alone."

Messrs. Haldane and Thomas[7] comment on exposure to high temperatures and sudden variations of temperature as follows:

"Miners are commonly exposed to high temperature underground, and to comparatively sudden cold on coming out, often with damp clothes on. Much stress has been laid on this fact, particularly in relation to lung diseases. Nevertheless, this cannot be an important cause of lung disease, for colliers are similarly exposed. Moreover, in England, colliers never wash and change their clothes at the pit-head, while Cornish miners almost invariably do so, in a heated building ('dry') provided on the mine. The effects of high underground temperatures on men and horses are certainly of considerable interest and economic importance, and we know from personal observations that in warm and moist air underground the body temperature often rises several degrees; but we can find nothing in the circumstances connected with the mode of occurrence of miners' phthisis to suggest that high underground temperatures are in any way connected with its causation."

Church writes[8] on the immediate results of high temperatures and hot vapors, as absent-mindedness, dizziness, fainting, vomiting, and, as graver results, insanity and death. His final conclusion is the following:

"The casualties positively traceable to the heat are therefore twelve per cent. of the whole. Probably the heat increases the bad effects of powder fumes and natu-

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[6] Comstock Mining and Miners, *Monograph IV.*, *U. S. Geological Survey*, p. 400 (1883).

[7] *Transactions of the Institution of Mining and Metallurgy*, vol. xiii., p. 383 (1903–04).

[8] Accidents in the Comstock Mines and Their Relation to Deep Mining, *Trans.*, viii., 95 (1879–80).

ral gases, and by making repairs to the shafts more frequently necessary it indirectly adds to the occasions when disasters may occur. I also confess to the belief, which is not sustained by observations upon specific casualties, that some allowance should be made for a less active mental condition, a dulling of the faculties, and a certain recklessness to which the heat sometimes goads the men. On the other hand the heat makes them more cautious except when under momentary impulses, and I have never seen American miners more careful of themselves than in these mines. On the whole the good and bad effects of the heat seem to nearly balance each other, and I think that an allowance of five per cent. for the casualties indirectly caused by the high heat would be sufficient."

The main facts brought out by these observers are that high temperatures and humidities cause, under some conditions, a notable rise in body-temperature; that all the bodily organs with the exception of the stomach are stimulated; that prolonged exposure to such conditions undoubtedly lowers vitality; that intermittent exposure produces no permanent ill-effects; that these conditions cannot be considered an important cause of lung-diseases; and, lastly, that under abnormal conditions loss of mental control and serious bodily disturbances result.

Local physicians say that the average working-life of the Comstock miner approximates 25 years, and that the miners do not show any greater susceptibility to any particular disease than the residents of the town. I am acquainted with miners who have worked more or less continuously underground for 30 years, and who are still capable of doing a good day's work. Compared with miners of other districts, the Virginia City miners may be said to be just as healthful, if not more so. The result is due in a large measure to the fact that the Virginia City miner observes certain precautions, and that the mine-managements provide the necessary facilities. Miners are careful not to expose themselves to cold drafts, since they work stripped to the waist. On passing from a hot to a cold place a heavy coat is used to protect the heated body. Wet clothes are either removed before going to the surface, or trousers and coat worn over them. All miners take hot and cold showers after coming off shift. Frequent drinking, and the bathing of the hands, wrists, arms, and head in ice-water are resorted to in all hot workings, and undoubtedly serve to keep down the body-temperature, while temporarily refreshing. Frequent rests are taken in special cooling-rooms, so placed in the workings as to receive the freshest and coolest air. In exceedingly hot work-

ings cold water from a hose is turned upon the miner while at work.

The effect of the underground conditions upon the blood was made the subject of a preliminary study by Prof. P. Frandsen at my request. His results are of importance, and in consequence are given in full in the accompanying note. His conclusion, given tentatively, is that "the conditions in the deep workings of the Virginia City mines do not have any particularly detrimental effects upon the composition of the blood. From a comparison of Tables XXVII. and XXIX. it appears that the main permanent effect is an increased hæmoglobin-content of the individual red blood-corpuscles."

Conditions are not favorable for a study of the comparative efficiency of miners working under normal temperatures and under those in the lower levels of the Comstock. The following conclusions are the results of my underground experience: Moderately high temperatures, from 95° to 105° F., with moderate humidities, from 50 to 70 per cent. relative humidity, and with air-currents of velocities from 200 to 300 ft. per min., do not prevent efficient work nor are they particularly uncomfortable.

A higher temperature, from 110° to 115°, together with the same conditions as above, decreases efficiency to a considerable extent.

A high temperature, from 110° to 115°, with high humidity and moderate velocity air-currents, very greatly impairs the miners' efficiency; and a still higher air-velocity, under the same conditions, renders workings more bearable, but miners cannot work very long at one time.

A moderately high temperature, from 95° to 105°, in a saturated atmosphere with no current, becomes very trying. Prolonged exposure with much exertion is dangerous.

A moderate temperature, from 90° to 98°, and saturated air-currents of a velocity of from 400 to 500 ft. per min., with more or less vitiated air, are conditions which are very trying and give a low labor-efficiency. Vitiated air will impair labor-efficiency to a greater extent than a high temperature.

X. Appendix.

By Prof. P. Frandsen.*

With a view to obtaining more exact information as to the physiological effects of the atmospheric conditions in the lower levels of the Virginia City mines, some preliminary studies were made upon the blood of subject No. 1 and several miners. To determine the effect of a temporary sojourn in the mines, blood of subject No. 1 was examined at Reno, Dec. 10, two days before going to Virginia City; Dec. 12, after remaining 2.5 hr. in the 2,000-ft. level of the Union shaft; on Dec. 13, after spending 4.5 hr. in the Sutro tunnel and south lateral; and on Dec. 14, at Reno. The temperatures in the workings visited ranged from 100° to 109°, with humidities of from 60 to 100 per cent. During both trips the subject and myself perspired profusely while underground, but felt no ill-effects other than a slight dizziness and a natural fatigue. The results of the examinations are given in Table XXVII.

Table XXVII.—*Subject No.* 1.

Date.	Place.	Red Corpuscles per c.c.	Leucocytes (white corp.) per c.c.	Hæmoglobin.	Hæmoglobin Index.
				Per Cent.	
Dec. 10...	Reno........................	5,160,000	7,111	95	92
Dec. 12...	Va. City, underground	4,820,000	12,333	...	...
Dec. 13...	Va. City, underground	6,496,000	12,277	100	77
Dec. 14...	Reno........................	6,220,000	9,888	95	76

The difference in red-counts on Dec. 10 and Dec. 12 may be ignored as too slight to have any significance. On Dec. 13, however, after the longer sojourn underground, there was an appreciable increase of red corpuscles, which is still to be noted on the following day. On both occasions there was a considerable leucocytosis, due mainly, as Table XXVIII. shows, to an increase in the number of neutrophiles.

Tables XXIX. and XXX. give the results of examinations of five miners who have been engaged in mining for 10 years or more, and for a year or more have been at work under the conditions described in this paper as characteristic of the deeper

* Professor of Biology, University of Nevada.

TABLE XXVIII.—*Differential Count of Leucocytes (White Blood-Corpuscles).*

Date.	Leucocytes per c.c	Percentage of Varieties				
		Lymphocytes.	Mononuclears.	Neutrophiles.	Eosinophiles.	Mast.
Dec. 10..	7,111	41.5	2.5	55.5	0.4	0.1
Dec. 12..	12,333	32.0	2.0	65.0	0.9	0.1
Dec. 13..	12,277	32.8	2.4	64.1	0.5	0.2
Dec. 14..	9,888	44.6	3.7	49.0	2 5	0.2

levels. All the subjects were in a fair state of health, and men of good physique, well nourished, and moderate users of tobacco and liquor. The examinations were made as soon as the men reached the surface at the close of their day's shift, and, except in the case of subject No. 2, before the shower which all take at the end of the day's work.

TABLE XXIX.—*Summary of Examinations of Blood of Miners.*

Subject.	Age.	Red Corpuscles per c.c.	Leucocytes per c.c.	Hæmoglobin.	Hæmoglobin Index.	Pulse.	Temperature.
				Per Cent.			
2	38	4,236,000	6,000	115	1.35	80	97.4
3	29	5,386,000	7,814	120	1.11	76	97.2
4	33	4,460,000	10,777	120	1.34	70	98.2
5	38	5,178,000	7,055	111	1.07	80	98.2
6	38	5,422,000	12,277	114.5	1.05	61	99.1

TABLE XXX.—*Differential Count of Leucocytes of the Blood of Miners.*

Subject.	Leucocytes.	Percentage of Varieties.				
		Lymphocytes.	Mononuclears.	Neutrophiles.	Eosinophiles.	Mast.
2	6,000	36.8	5.8	56.6	0.4	0.4
3	7,814	52.4	3.2	48.5	0.5	0.4
4	10,777	40.7	3.5	54.0	1.0	0.8
5	7,055	53.3	2.3	43.1	0.8	0.5
6	12,277	34.3	3.7	59.1	2.8	0.7

The most striking feature revealed by Table XXIX. is the uniformly high hæmoglobin index of the miners' blood. The

percentage of hæmoglobin was determined by means of Dare's instrument. In two cases only is the number of red corpuscles below the normal average, 5,000,000 for persons at sea-level; but the reduction is hardly great enough to be considered indicative of anæmia. Moreover, microscopic examination of stained preparations showed no abnormal appearances. The red corpuscles were of uniform size and shape, and chiefly noteworthy for their richness in hæmoglobin, indicated by the intensity and uniformity with which they took the eosin stain. The foreman said of subject No. 2 that he did not stand the heat as well as the other men, and had fainted on several occasions.

Except possibly in the cases of subjects Nos. 1 and 6, the leucocyte-counts are within the limits of variability in normal individuals. Most striking is the high percentage of lymphocytes, particularly in subjects Nos. 3 and 5.

As living in high altitudes is claimed by many authorities to bring about a marked increase in the number of red corpuscles over the number normal to residents of the sea-coast, I give in Table XXXI. for comparison with Tables XXVI. to XXX. the results of counts made upon subjects resident in Reno and Virginia City, all of whom were in good health and had resided in the locality designated for a number of years.

TABLE XXXI.—*Summary of Blood-Examinations in Relation to Altitude.*

Subject.	Age.	Red Corpuscles per c.c.	Leucocytes per c.c.	Hæmoglobin.	Hæmoglobin Index.	Place.	Altitude.
				Per Cent.			Ft.
1	32	5,160,000	7,111	95	92	Reno.	4,553
7	32	5,399,000	8,944	105	97	Reno.	4,553
8	33	5,536,000	8,944	109	98	Va. City.	6,000
9	40	6,074,000	6,777	104	85	Reno.	4,553
10	20	6,378,000	4,111	110	86	Reno.	4,553

Subject No. 9 for several years has been making regular and frequent trips to the summit of Mount Rose, elevation 10,800 ft., and is an experienced mountain-climber. The examinations of both Nos. 9 and 10 were made about 24 hr. after their return from a fatiguing three days' trip to the summit of this mountain.

These counts do not show as great an increase in the number

of red corpuscles as the published accounts of other observers would lead us to expect in these altitudes. In comparison with these results, the number of red corpuscles obtained in the counts on the miners would not seem to be very far from the normal. Differential leucocyte-counts were made on the blood of these subjects, as given in Table XXXII.

TABLE XXXII.—*Differential Count of Leucocytes in Relation to Altitude.*

Subject	Leucocytes per c.c.	Percentage of Varieties.				
		Lymphocytes.	Mononuclears.	Neutrophiles.	Eosinophiles.	Mast.
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent
1	7,111	41.5	2.5	55.5	0.4	0.1
7	8,944	33.4	4.5	57.6	3.5	1.0
8	8,944	32.2	5.4	60.4	1.7	0.3
9	6,777	44.4	5.2	49.2	1.0	0.2
10	4,111	42.0	5.5	50.0	2.1	0.4

The only point to be noted is the relatively high percentage of lymphocytes.

While these studies are too few to warrant an extended discussion or the statement of any very definite conclusions, they are of interest as indicating that the atmospheric conditions in the deep workings of the Virginia City mines do not have any particularly detrimental effects upon the composition of the blood. From a comparison of Tables XXIX. and XXXI., it appears that the main permanent effect is an increased hæmoglobin-content of the individual red corpuscles.

ACKNOWLEDGMENTS.

In collecting the foregoing data I received many courtesies from the officials of the Ophir mine, the Sutro Tunnel Co., the Yellow Jacket, the Caledonia, and the Overman Mining Companies. I wish especially to acknowledge the assistance of Dwight T. Smith, of the Virginia Mining School; T. McCormick, superintendent of the Ophir mine; T. Sullivan, of the Ophir mine; B. O'Hara, foreman of the Sutro tunnel; and Leon M. Hall, of the Ward Shaft Pumping Association. To Professor Frandsen, for his contribution, I am particularly grateful.

Electric Mine-Hoists.*

BY D. B. RUSHMORE AND K. A. PAULY, SCHENECTADY, N. Y.

(Pittsburg Meeting, March, 1910.)

I. Introduction.

Of primary importance in mine-installations is the hoist, which has a very direct bearing on the successful operation of a mine. Conditions vary greatly with different mines, and especially in different localities. Such factors as depth, incline, the number of levels, permissible or desirable speeds, conditions of ore, etc., are always more or less special in each case. Veins of ore are never exactly duplicated, and the nature of the ground through which shafts are sunk may considerably modify permissible values. As mining-laws are made by the different States, they necessarily vary somewhat, and, even when not fully observed, they introduce factors which qualify the conditions of hoisting men and ore. The amount of timbering required is often of importance as relating to hoisting-conditions. Methods of loading ore affect the time required, as also does the question of the use of cars or skips. Safety-precautions must be very carefully considered, and the number of men in each mine, the number of compartments, and often the method of removing water from the mine, must have careful consideration.

While a general discussion of the subject of hoisting is possible, most cases are entirely special, and can be considered only in connection with the peculiar conditions pertaining to that particular installation.

The cost of installation of the hoisting-plant may be an appreciable amount, while the cost of raising the ore may be but a small part of the total operating-charge. In many cases, however, the output of the mine is limited by the capacity of the hoist, and the latter thus becomes of the first importance.

* Presented also, by mutual agreement, at the meeting of the American Institute of Electrical Engineers, New York, N. Y., Mar. 11, 1910.

Where shafts have not been sunk to their final depths, the conditions of operation are of necessity constantly changing, and it is impossible to predetermine with exactness the precise conditions of operation which will be followed in practice.

1. *Power for Mines.*—Power is used for drilling, tramming, pumping, ventilating, hoisting, compressing air, crushing rock, and for many minor operations. In coal-mines, the washeries and breakers, and in metal-mines, the mills and concentrators, are ordinarily located in proximity to the shafts. The problem of lighting always exists.

Owing to the distances between different points of applications of power, not only the question of utilization, but also that of transmission, becomes of importance. Three forms of power—steam, compressed air, and electricity—are to be considered. Originally, of course, the power must come from coal or water-power.

2. *Choice of System.*—The choice of the best system of hoisting in any particular case is the result of considering carefully many different factors. Most important among these is the cost of operation and installation. In this regard the location of the power-house to insure the best and cheapest supply of fuel and water is of primary importance. It is highly desirable to group a large number of mines, so that they may be supplied from one power-station. As a rule, mining-shafts are not well situated as regards the supply of coal and water, so that it is usually necessary to transmit power for some distance, and this is best done by electricity. The problem then becomes one of the utilization of power, or the generation of electricity by means of steam-turbines or gas-engines. In metal-mining, fuel is usually expensive, and often but little water is available at the mines, so steam hoisting-engines are generally run non-condensing. Where the reverse is the case, it is in most cases cheaper to transmit electricity to the mines than to pump condensing-water there.

3. *Efficiency of Steam Plants.*—Steam hoisting-plants are known to be very inefficient, but the exact figures are usually difficult to obtain. With non-condensing engines and an extremely intermittent load on both engines and boilers, the economy necessarily is very poor. Steam-engines must be designed for starting-conditions, where they take steam under

full stroke, and this necessitates their running with an early cut-off when hoisting. With a number of plants close together, there is no way of returning power to the line or of smoothing the peaks of the load. It is impossible, when a steam-engine is used, to store power in retardation. There is also a limit to the depth at which steam-engines can be satisfactorily placed, and their installation in mines is thus very undesirable.

4. *Advantages of the Electric System.*—In many cases the electric system of hoisting has advantages which give it decided preference. The power-house may have the most favorable location—power may be taken from some existing transmission-system, or a water-power may be developed for the purpose. Power may be centrally generated at the highest efficiency, and distributed over a large area. Electricity is most easily applied to all work on both the surface and the interior of mines. One of its greatest advantages in practice is the ease of making extensions to the development. With one station and many individual loads, an overlapping of peaks occurs, and a consequent reduction in boiler- and generating-capacity is effected. The cost of installation and operation is much reduced, a much improved load-factor results, and fewer operators are required.

For underground pumping and tramming, and where the hoists are located in the mine, electricity has every advantage. For use with electric hoists, safety-devices have been developed which prevent overwinding and which also limit the acceleration. Power can be returned to the system in braking and in lowering unbalanced, and a much higher fuel-economy can be obtained.

The use of hoists operated by compressed air has long been considered, and at present some installations are being made. With the usual features of such equipment, it is necessary to cool the air during compression and to reheat it before use. In general, serious questions would arise concerning the efficiency of any system using compressed air for hoisting. The efficiency of an electric hoisting-system is not open to question, and can be figured with exactness. There is no reason why advocates of compressed-air systems should not be required to give the same guarantees and to state positively just what efficiency they are sure of obtaining.

II. Motor Characteristics.

Large electric mine-hoists are almost universally driven either by shunt-wound direct-current motors or polyphase induction-motors, the characteristics of which especially adapt them to meet the peculiar conditions imposed. While in many of their characteristics these two types of motor are similar, they differ widely in others, which are of more or less importance, depending upon special conditions of individual cases.

Fig. 1 gives the efficiencies, currents, and speeds of the direct-current shunt-motor at various loads for constant impressed voltage, and Fig. 2 gives similar curves for the induction-motor, to which is added the power-factor curve. By reference to these curves it will be seen that the free-running speed of each motor is limited, and that the variations in speed with changes in load are small; that each becomes a generator when driven above speed, and may therefore be used as a brake when lowering unbalanced loads, returning power to the supply-system; that the efficiencies of both when operating either as motor or generator are virtually the same for corresponding loads.

The speed of the shunt-motor for a given load may be varied between standstill and full speed either by changing the potential of the supply-system, or by inserting resistance in series with its armature. However, because of the inefficiency of this latter method of control, it is seldom, if ever, used in connection with large hoist-motors. The only practical method of obtaining a similar variation in the speed of an induction-motor is by changing the amount of resistance connected in its armature circuit. Fig. 3 shows the efficiencies of the shunt-motor at various speeds when exerting full-load torque, the variations in speed being obtained by voltage-control, and the efficiencies of the induction-motor being under similar conditions, except that in this case the variations in speed are obtained by armature rheostatic control. By reference to the curves it will be seen that for a given torque, the efficiency-curve of the shunt-motor at reduced speeds resembles that for the constant-speed motor at reduced loads, while the efficiency-curve of the induction-motor is a straight line between full-load efficiency and speed, and zero efficiency and speed. From this it follows that, for a given value of torque, the input to the shunt-motor at reduced

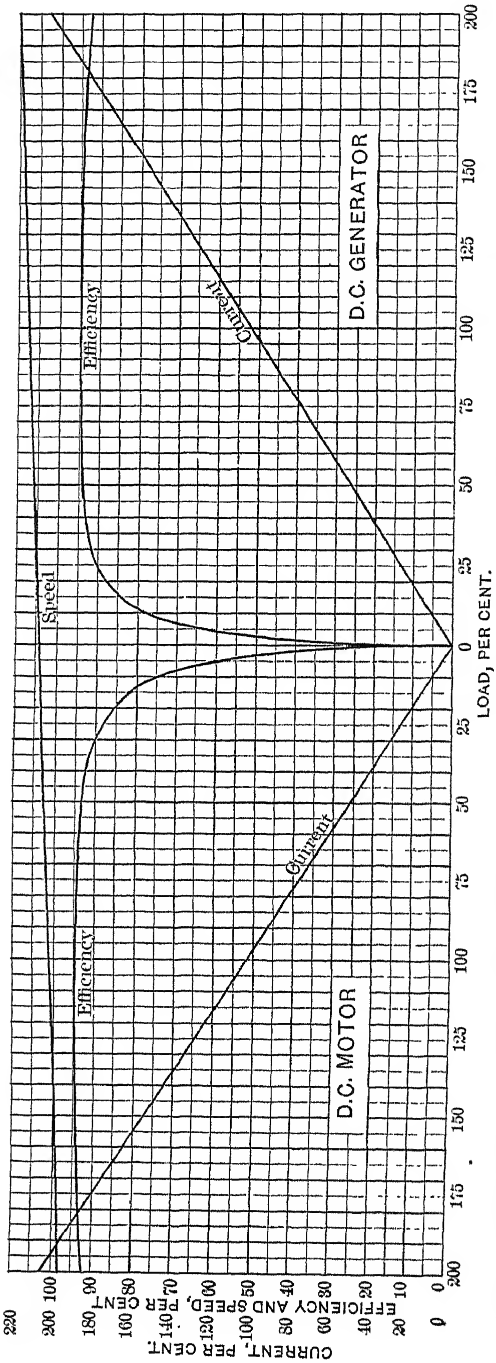


FIG. 1.—EFFICIENCY-, CURRENT-, AND SPEED-CURVES OF DIRECT-CURRENT SHUNT-MOTOR AT VARIOUS LOADS FOR CONSTANT IMPRESSED VOLTAGE.

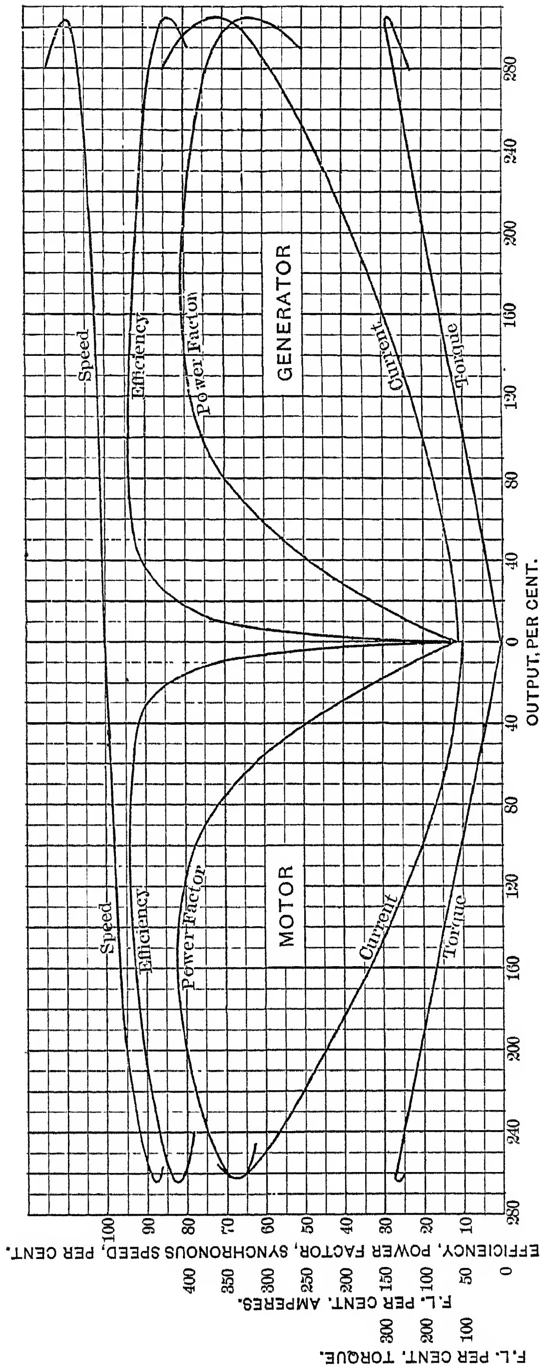


Fig. 2.—Efficiency-, Current-, Speed-, and Power-Factor-Curves of Induction-Motor at Various Loads for Constant Impressed Voltage.

speeds, except for very low speeds, is approximately proportional to the speed, while the input to the induction-motor is constant and independent of the speed. It will also be noted that the shunt-motor may be driven as a generator at reduced speeds, while the induction-motor can be made to generate only when driven above synchronous speed. Where the conditions are such that it is desirable to drive the hoist at reduced speeds for any considerable lengths of time, the efficiency of the induction-motor drive may be improved by using a motor designed for two speeds, or by using a concatenated set, but the advan-

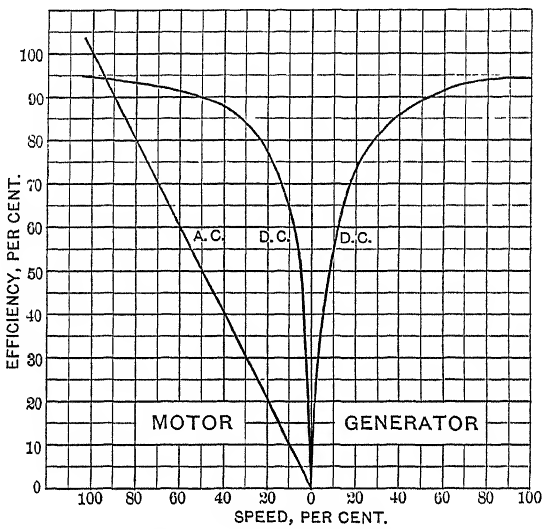


FIG. 3.—EFFICIENCY-CURVES OF SHUNT-MOTOR AT VARIOUS SPEEDS.

tages gained are seldom sufficient to off-set the increased first-cost and the necessary complication of the control.

The number of poles, and therefore the diameter, of an induction-motor is determined by its speed and the frequency of the supply-system, the number of poles varying inversely as the speed for a given frequency, while the frequency of the electromotive force generated in the armature of a direct-current motor is independent of the supply-system. While this is of little importance in the designing of motors of moderate speed for gearing, it permits of a better proportioning of the length and diameter of shunt-motors of very slow speed than is possible in the case of similar induction-motors for direct connection, especially where the frequency of the supply-system is 60 cycles.

As pointed out, the efficient speed-control of the shunt-motor is only obtained by varying the voltage of the supply-system, the usual method being to provide a generator for each motor and to vary the generated potential. As mine-shafts are usually scattered over a considerable area, and the conditions in close proximity to the shafts are not such as to permit of the economical generation of electric power, the central electric station is usually placed at a considerable distance from the hoists. The power is generated and transmitted to the mines as alternating current, and is then transformed at each shaft into direct current by motor-generator sets. The losses caused therein must be charged against the shunt-motor when compairing its efficiency with that of the induction-motor, which may be connected either directly or through highly-efficient static transformers with the alternating-current distributing-system. The torque and current for the two types of motor are approximately proportional within their operating-limits.

III. Hoist-Load Diagrams.

Before discussing the various systems of electric hoisting it will be well to consider the nature of a mine-hoist load. Mine-hoists may be divided into six types, depending upon whether the rope is wound on a reel, a cylindrical drum, a conical drum, a cylindro-conical drum, Whiting drums, or carried over a Koepe disk, the choice of any particular type depending largely upon the depth of shaft, the maximum permissible hoisting-speed, the location of the hoist with respect to the shaft, the number of levels which are being worked simultaneously, and whether or not the shaft-conditions permit the use of a tail-rope.

The hoists are generally operated in balance; that is, the weight of the skip (or cage and car, as the case may be) carrying the ore is balanced by a similar empty skip, which is lowered in a second compartment simultaneously with the hoisting of the loaded skip in the first, the loaded skip being dumped at the top and the empty one loaded at the bottom, and the cycle then repeated. To permit of adjustment of the length of the ropes for hoisting from different levels, it is customary to use two reels or drums mounted on the same shaft, one being keyed to the shaft and the other being driven by it

through some form of clutch. The length of rope on the Koepe disk-hoist or the Whiting hoist cannot conveniently be adjusted for different levels except within very small limits.

For the purpose of comparing the load-diagrams of the different types, each hoist is assumed to lift 8,000 lb. of ore in a skip weighing 5,000 lb., from a vertical depth of 2,500 ft., at an average speed of 2,000 ft. per minute, allowing 20 sec. for acceleration and 15 sec. for retardation. The rope for the reel-hoist is assumed to be 0.5 by 5 in., weighing 4.5 lb. per foot, and that for the others $1\frac{3}{8}$-in. round steel rope, weighing 3 lb. per foot.

1. *Reel-Hoist.*—As its name indicates, the reel-hoist consists of two large reels on which the rope supporting the skips is wound in a spiral, the distance between the flanges of the reels being approximately equal to the width of the flat rope used. The reel-hoist is generally used where it is necessary to place the hoist very close to the shaft. As the minimum diameter of the reel, usually from 5 to 8 ft., is limited, and as its maximum diameter is determined by the thickness and length of the rope, the depth from which hoisting may be done at a given average speed by reel-hoists is governed by the maximum permissible hoisting-speed.

Let us assume that the loaded skip is at the bottom of the shaft and that the empty skip is at the top. Then the length, L_a, of the ascending rope with the skip at any point in the shaft may be obtained from the equation

$$L_a = D - 2\pi a r_1 - \pi a^2 b,$$

and its moment, M_1 about the drum-shaft by

$$M_1 = (D - 2 a \pi r_1 - \pi a^2 b)\, r n \cos\phi,$$

in which D = depth of shaft; b = thickness of the rope; n = the weight of the rope per foot; r_1 = the radius of the rope on the reel when the skip is at the bottom of the shaft; a = the number of turns of the reel in raising the skip from the bottom of the shaft to the point in question; $r = r_1 + a b$ = the radius of the rope on the reel after a turns; and ϕ = the angle which the shaft makes with the vertical.

The moment of the ascending load is obtained from the equation

$$M_2 = (W_1 + W_2) r \cos \phi,$$

where W_1 = the weight of the skip (or cage and car, as the case may be); and W_2 = the weight of the ore.

Plotting M_1 and M_2 against revolutions of the reel, we obtain curves M_1 and M_2 in Fig. 4.

The moment M_1' of the descending rope may be plotted from the values obtained for the ascending rope, by simply assuming the center of co-ordinates in Fig. 4 transferred from the left to the right side of the curves, turn No. 10 of the descending load corresponding to turn No. 81 of the ascending load, etc.

Similarly, the moment M_2' of the descending load may be obtained by substituting W_1' for $W_1 + W_2$ in the equation for M_2, and plotting as directed for the descending rope.

Moments M_1' and M_2' are plotted below the reference-line, since their tendency to rotate the reel is opposed to that of M_1 and M_2.

Denoting the moment of the total friction and windage by M_f, the resultant moment M_0 of the ascending ore-skip, rope, and friction, and the descending skip and rope, is expressed by $M_0 = M_1 + M_2 - M_1' - M_2' + M_f$, which is the moment of the force, or the torque which must be applied to the reel to raise the ore.

The moment M_f of the friction is extremely difficult to obtain, and varies considerably with local conditions, but it is usually assumed to be approximately 15 per cent. of the average value of $M + M_2 - M_1' - M_2'$.

During the period of acceleration, a force additional to that required to raise the load and overcome friction must be applied to the reel for accelerating the reels, ropes, etc. Assuming a uniform rate of acceleration, the moment M_3 of the force necessary for accelerating the ascending load, rope, and one reel and clutch, may be determined from the equation.

$$M_3 = \frac{\Sigma\, W S}{g\, t_a} p,$$

where $\Sigma\, W$ = the sum of the weights of the skip, ore, rope, one reel and clutch, and sheave, reduced to a common radius

of gyration, p; $S =$ the speed at the end of the radius of gyration in feet per second at the end of acceleration; $g = 32.2$, and t_a the time of acceleration in seconds.

Similarly, the moment M_3' of the force required for accelerating the descending skip, rope, etc., may be found from the equation

$$M_3' = \frac{\Sigma W' S}{g\, t_a} p,$$

where $\Sigma W' =$ the sum of the weights of the descending skip, rope, reel, and sheave, reduced to the radius of gyration, p.

$$S = 2 \pi p R,$$

where $R =$ the revolutions of the reel per second at full speed.

When the time allowed for making one complete trip is given, R may be found from the following equation

$$R = \frac{-r_1 + \sqrt{r_1^2 + \dfrac{b L}{\pi}}}{b} \times \frac{1}{T - 0.5\,(t_a + t_r)}$$

Where $T =$ total time of lift in seconds (not including time for loading or dumping),

$t_a =$ time for acceleration in seconds,

$t_r =$ time for retardation in seconds.

At the end of the cycle all of the energy stored in the revolving parts of the hoist and its load is returned as the load is brought to rest. The moments M_4 and M_4' of the retarding-forces may be found from the expressions for M_3 and M_3', respectively, substituting for ΣW and $\Sigma W'$ the corresponding weights at the beginning of retardation.

Throughout the cycle there is a gradual increase in the speed of the ascending ore-skip and unwound rope, and a similar decrease in the speed of the descending skip and unwound rope, but the accelerating- and retarding-forces are small and their moments may be neglected.

The resultant moments M_A and M_R during the periods of acceleration and retardation respectively are expressed by the equations

$$M_A = M_1 + M_2 + M_3 + M_3' + M_F - M_1' - M_2' ;$$

$$M_R = M_1 + M_2 + M_F - M_1' - M_2' - M_4 - M_0'.$$

M_A, M_0 M_R of Fig. 4 is the resultant moment-diagram for balanced hoisting under the conditions assumed.

While hoists are generally operated in balance, it is frequently necessary to run them unbalanced for short periods while repairs are being made. The moment-diagram $M_A M_0 M_R$ for unbalanced hoisting is obtained from the equations

$$M_A = \frac{M_F}{2} + M_1 + M_2 + M_3;$$

$$M_0 = \frac{M_F}{2} + M_1 + M_2;$$

$$M_R = \frac{M}{2} + M_1 + M_2 - M_4.$$

The speed of the ascending or descending skips in feet per minute at any point may be obtained from the equation

$$V = 120 \pi r R.$$

2. *Cylindrical-Drum Hoists.*—The cylindrical-drum hoist is the type almost universally used for comparatively shallow shafts, and very frequently for the deeper ones. It consists of two cylindrical drums, upon which the rope is wound in one or more layers, the diameters of the drums varying from 5 or 6 ft. to 25 feet.

The general equations for determining the several moments which make up the reel-moment diagram are made applicable to the cylindrical-drum hoist by simply making b equal to zero.

For the cylindrical drum,

$$R = \frac{D}{2 \pi r} \times \frac{1}{T - 0.5 (t_a + t_r)}.$$

The moment-diagrams for the cylindrical-drum hoist are shown in Fig. 5. The notches in the diagram are due to the increase in diameter of the drum with each layer of rope.

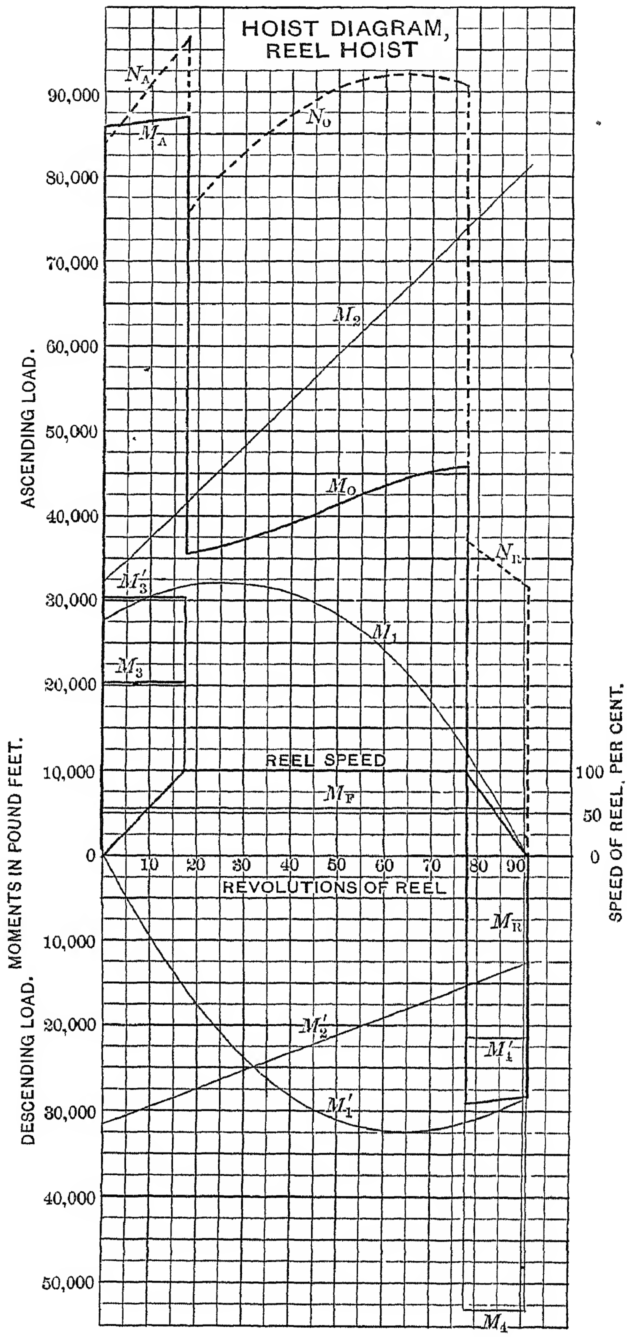


FIG. 4.—LOAD-DIAGRAM, REEL HOIST.

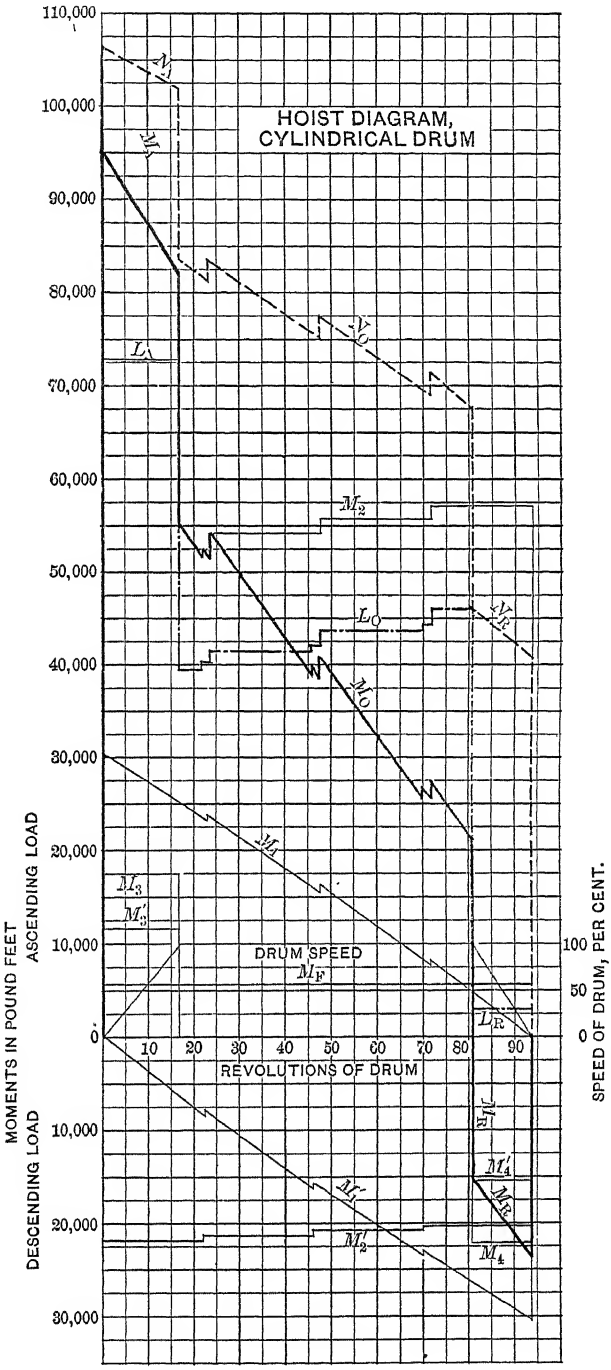


FIG. 5.—LOAD-DIAGRAM, CYLINDRICAL-DRUM HOIST.

By reference to the figure, it will be seen that M_0, the resultant moment of the ore-skips and ropes, is very large at the first part of the cycle and decreases very rapidly towards the end of the cycle, this being due to the influences of M_1 and M_1', the moments of the rope. This difference in M_0 at the beginning and at the end of the cycle increases with the depth of the shaft, the weight of ore per trip remaining constant; or for the same depth of shaft with reductions in the weight of ore hoisted, M_0 often becoming partly zero or negative towards the end of the cycle. The harmful effect of this extreme variation in M_0 is two-fold. First, the engine must be larger than otherwise necessary, in order to start the hoist, and second, it operates at an inefficient cut-off at the end of the cycle. To reduce this variation in M_0 at the beginning and at the end of the cycle, the conical drum has been introduced.

3. *Conical-Drum Hoists.*—In the conical-drum hoists, the ropes are wound in single layers on two large cones, the rope being wound from the small to the large end of the cone. The method of determining the moment-diagrams is the same as for the reel-hoist, making b equal to the increase in the radius of the cone for one turn of the rope. By reference to Fig. 6, which shows the moment-diagrams for the conical-drum hoist, it will be noted that the unbalancing due to the rope in the shaft has been entirely compensated for; in fact, M_0 actually increases towards the end of the cycle. By varying the angle of the cone, M_0 may be made to increase or decrease towards the end of the cycle, or remain practically constant.

4. *Cylindro-conical Drum Hoist.*—The use of the conical drum, as was the case with the reel, is limited to comparatively shallow shafts. For depths below which the use of the conical drum is impracticable, it is necessary to compromise, using a drum which, as its name indicates, is a combination of cone and cylinder. The rope is wound from the small end of the cone over the conical part of the drum in a single layer; then on to the cylindrical portion in one or more layers, depending upon the length of the rope. The load-diagram is readily obtained by dividing the cycle into two parts, and treating the conical and cylindrical portions of the drum separately, determining the moments over the conical part as directed for the conical-drum hoist, and over the cylindrical portion as directed for the cylin-

drical-drum hoist. The moment-diagram, Fig. 7, shows very clearly the effect of the conical portion of the drum, although the improvement in shape of the moment-curve, over that for the cylindrical drum, is not so marked as it is for greater depths of shaft.

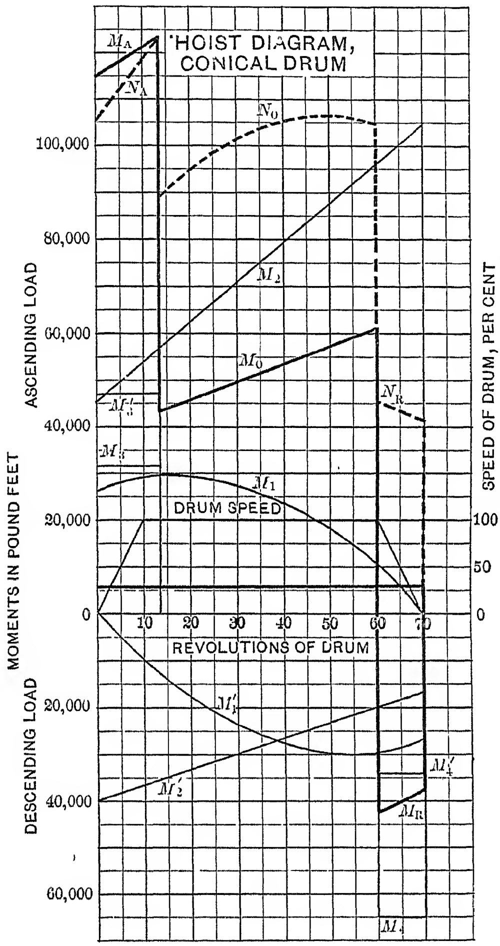


FIG. 6.—LOAD-DIAGRAM, CONICAL-DRUM HOIST.

5. *Tail-Rope.*—The skips or cages are sometimes connected by a tail-rope, which passes over a sheave at the bottom of the shaft, thus making the total weight of the ascending and descending ropes the same, independent of the location of the cages in the shaft. The effect of the addition of the tail-rope is shown in Fig. 5, L_A, L_O L_R being the moment-diagram re-

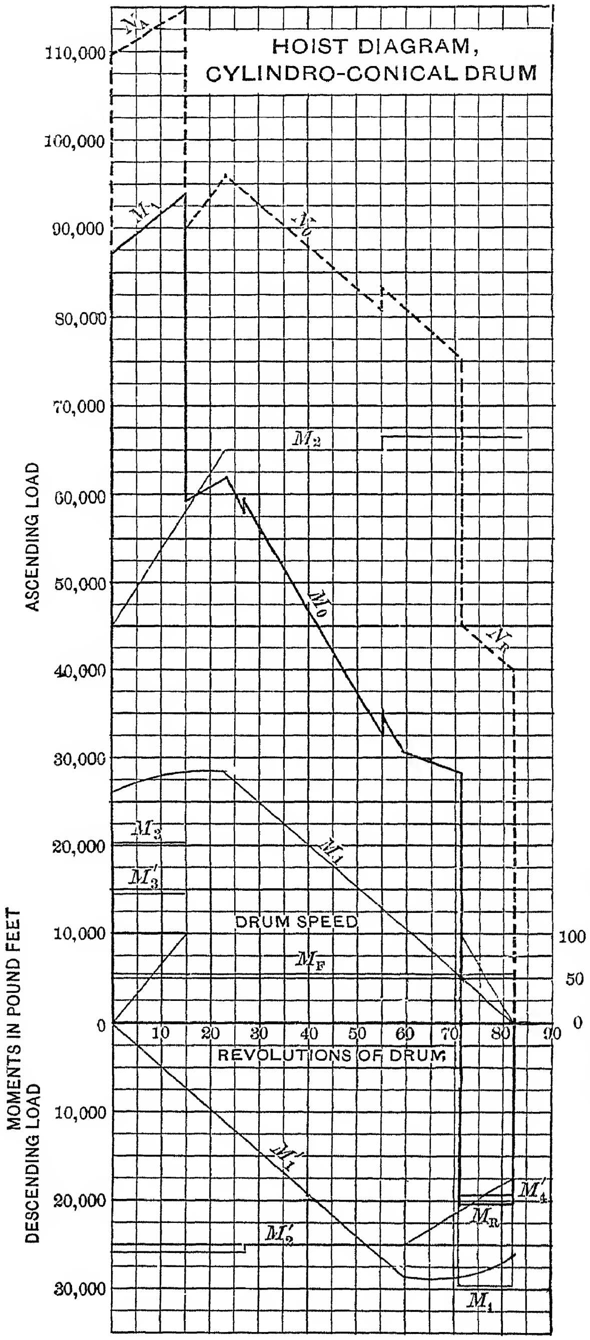


FIG. 7.—LOAD-DIAGRAM, CYLINDRO-CONICAL DRUM HOIST.

sulting from the addition of a tail-rope to the cylindrical-drum hoist, which otherwise remains unchanged.

6. *Koepe Disk-Hoist.*—A type of hoist very common throughout Europe, but which has never been installed in America, is that known as the Koepe disk-hoist, which consists simply of a large grooved wheel over which the rope passes once, the friction between the disk and the rope being sufficient to move the cages or skips in the shaft. Hoists of this type are always operated in balance and a tail-rope is used with them. They are not well adapted for hoisting from several levels, because of the fixed position of the cages or skips, which can be changed only with difficulty to correspond with different levels. The moment-diagram of this type of hoist is similar to that of the cylindrical-drum hoist with a tail-rope, except that b_0 will be a horizontal line, the notches due to the different layers of rope on the drum not being present in the Koepe disk-hoist diagram.

7. *Whiting Hoist.*—In order to increase the arc of contact between the rope and the wheel, Mr. Whiting substituted two narrow drums for the Koepe disk, the drums being placed one directly in front of the other, and the rope being passed four or five times over both drums. For the purpose of taking up the stretch in the rope and making small adjustments of the cages, one side of the rope is carried back over a movable sheave mounted on a carriage resting on rails, the adjustments being made by changing the position of the carriage. Whiting hoists are always operated in balance and seldom, if ever, without a tail-rope. The moment-diagram is exactly similar to that of the Koepe disk-hoist.

The moment-diagrams may be transformed into horse-power-time diagrams by the use of the equation

$$\text{horse-power} = \frac{2\,\pi\,M\,R}{550},$$

where $M =$ the moment in pound-feet and $R =$ revolutions of the drum per second.

The horse-power diagrams corresponding to the moment-diagrams of Figs. 4, 5, 6, and 7 are shown in Fig. 8, the curves for the balanced hoisting only being shown. Attention is called

to the difference in the heights of the peaks during accleration, the magnitude of these peaks having an important influence

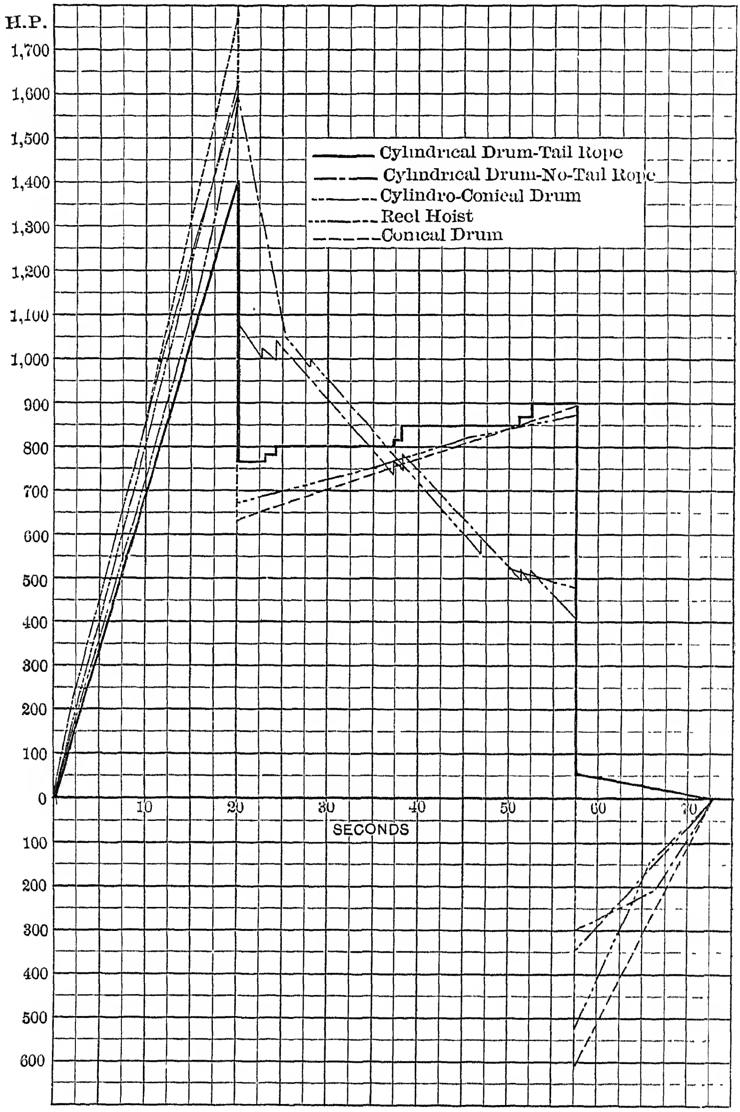


FIG. 8.—HORSE-POWER DIAGRAM FOR BALANCED HOISTING.

on the design of a motor for driving the hoist, its efficiency for the complete cycle, and the cost of power, if power is purchased.

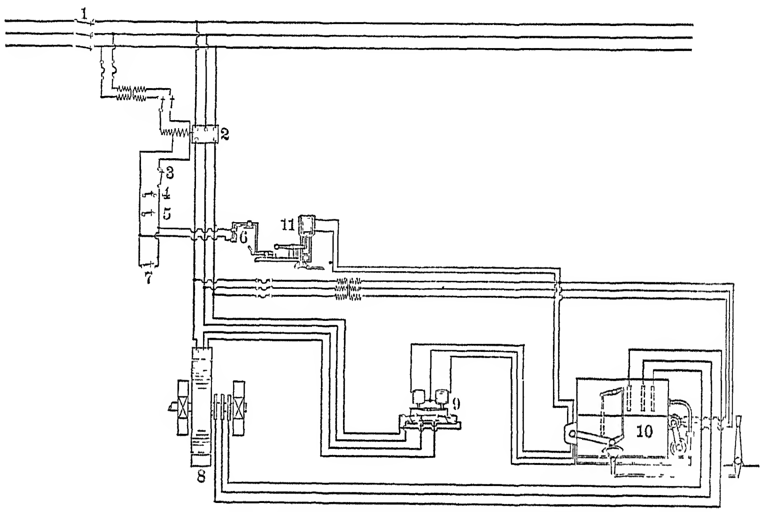
IV. Systems of Electric Hoisting.

The early application of electric power to mine-hoisting was confined to small hoists, moving cars on inclines, or hoisting light loads vertically from comparatively short depths. The complete success of these early installations led to the use of electric motors for winding from greater depths, until to-day ore is hoisted by electric power from some of the deepest mines in the world. The use of electric motors for driving the hoists permits of the substitution of a large central electric generating-station, which operates at a comparatively high load-factor, and which is placed where the conditions are more favorable for the economical development of power than they are at the mines, in place of isolated steam-plants located at the shafts, where condensing-water is seldom available and where fuel is often expensive. The speed of the hoist-motor when lowering unbalanced loads is automatically limited to approximately the hoisting-speed. Safety-devices in the nature of limit-switches can readily be applied to prevent overwinding, and the acceleration of the hoist is made automatic; all of which tend to minimize the possibility of accident resulting from carelessness on the part of the operator. Not only is the speed limited when lowering unbalanced, but a large part of the energy, which with the steam-driven hoist is absorbed by the brakes, is returned to the electric supply-system, thereby improving the economy of operation and reducing the wear on the mechanical brakes.

Many systems of electric hoisting have been proposed, each with the view of meeting some peculiar condition, or eliminating some real or apparent objection in the others, but virtually all the installations are confined to four systems, shown in Figs. 9, 12, 15, and 20. To assist in the comparison of the several systems, we have assumed a hoisting-cycle, for both balanced and unbalanced operation, and have calculated the current and power taken by each system under the conditions assumed.

The first and simplest system is shown in Fig. 9, and consists of a polyphase induction-motor, direct connected or geared to the hoist-drum. The speed of the motor is controlled by a variable resistance in its rotor circuit, which, because of the magnitude of the currents involved, is usually some form of

water-rheostat. A common type of water-rheostat consists of a tank, usually of boiler-plate riveted together, and divided into two compartments: one the rheostat proper, and the other a cooling-tank. The electrolyte is pumped from the cooling-tank into the rheostat proper, entering at the bottom of the rheostat and flowing out over the top of an adjustable weir, back into the cooling-tank. The resistance in the rotor circuit is varied by changing the height of the electrolyte in the rheostat proper by means of the adjustable weir. The electrodes are usually thin iron plates hung on insulators, all phases



1, 2. Line-switches. 3, 4, 5, 6, 7. Safety-switches for preventing overwinding and stopping the hoist. 8. Hoist-motor. 9. Reversing-switch. 10. Liquid controller. 11. Automatic brake.

FIG. 9.—INDUCTION-MOTOR SYSTEM.

being in the same compartment. At least one electrode per phase is of extra length, extending below the lowest level of the liquid, in order to prevent the rotor circuit from being opened. The most common form of electrolyte is a simple salt solution. The control of the rheostat is by means of a lever located on the operating-stand.

Figs. 10 and 11 give the current- and power-curves for one complete balanced and unbalanced cycle, respectively. Frequently the cage is moved a few feet only, to obtain a proper setting of the cage or skip. The power taken for such short movements is shown by the right-hand curves of Fig. 10. In

these curves, as well as in those which apply to the other systems, power delivered to the hoist is shown above the reference-line, and that returned by the hoist, below it.

By reference to the curves, it will be noted that the horse-power and current taken by the motor are constant during the period of acceleration; that the efficiency for this period is very low, approximately 45 per cent.; that no power is returned to the supply-system during the period of retardation; and that

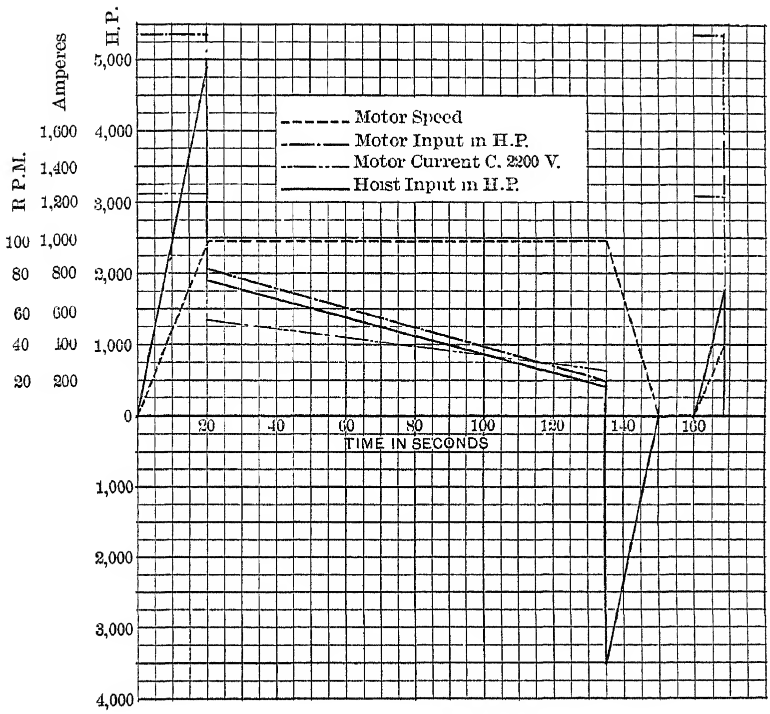


FIG. 10.—CURRENT- AND POWER-CURVES OF A BALANCED CYCLE OF THE SYSTEM SHOWN IN FIG. 9.

the power-consumption for small movements of the cage or skip is very large. On the other hand, the efficiency during the period when the hoist is running at full speed is high, approximately 90 per cent., and no power is consumed while the hoist is at rest.

The efficiency over the complete cycle obviously decreases rapidly with a decrease in the time during which the hoist is driven at full speed. It follows from this that when hoisting is

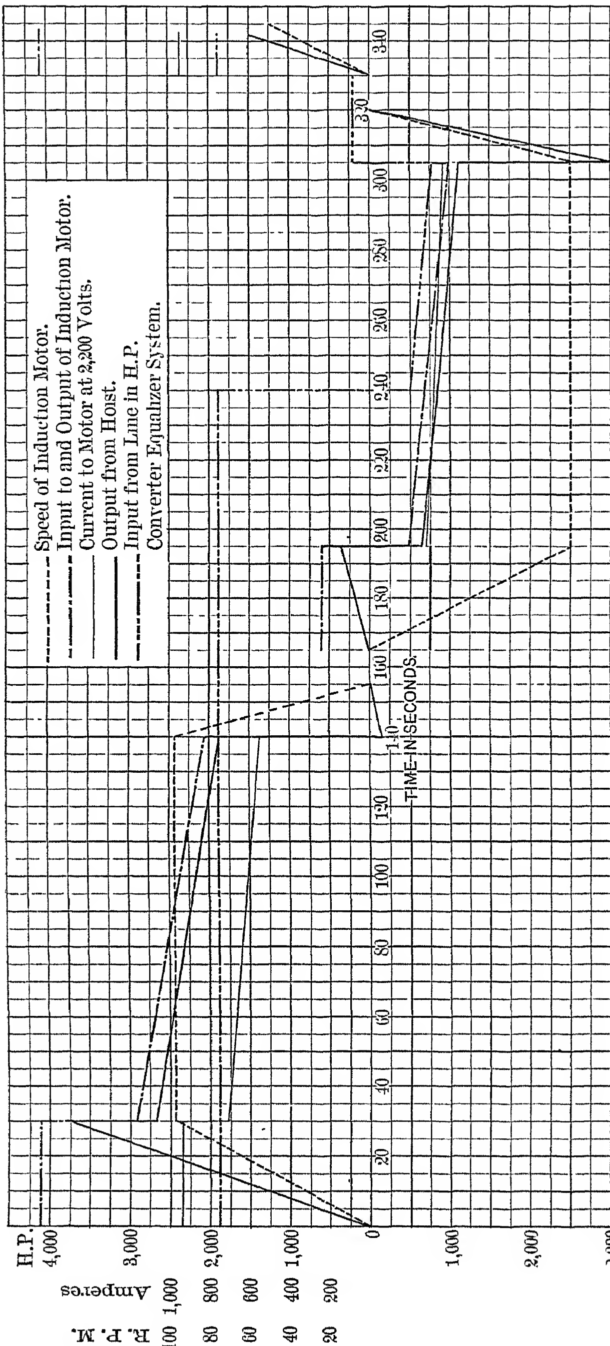
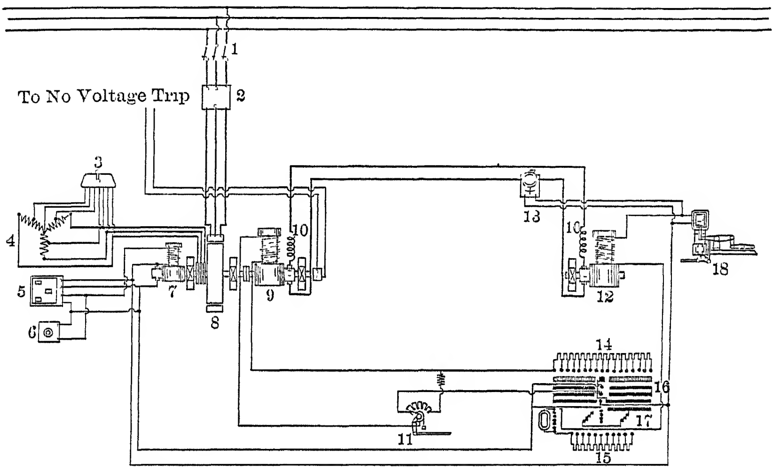


FIG. 11.—CURRENT- AND POWER-CURVES OF AN UNBALANCED CYCLE OF THE SYSTEM SHOWN IN FIG. 9.

to be done from several levels, the efficiency at the maximum depth alone cannot be used as a basis for comparing the hoist driven by the induction-motor with other systems. The efficiency of the cycle increases with an increased rate of acceleration, from which it follows that an induction-motor for hoisting should be designed for a high maximum output to permit of a rapid acceleration.

The power returned to the system when lowering the empty skip unbalanced is shown by the right-hand curves of Fig. 11.



1, 2. Line-switches. 3, 4. Controller and resistance, induction-motor. 5, 6. Tirrill regulator and rheostat for exciter. 7, 8, 9. Motor-generator set. 10. Commutating-pole and compensating field winding. 11. Safety-device. 12. Hoist-motor. 13. Circuit-breaker. 14, 15, 16, 17. Rheostat and controller for generator and hoist-motor. 18. Automatic brake solenoid.

FIG. 12.—SYNCHRONOUS MOTOR-GENERATOR SYSTEM.

A comparison of the power taken by the motor in hoisting the loaded skip with that returned when it is lowered empty, shows that approximately 20 per cent. of the power taken for hoisting is returned in lowering.

The second system is that shown in Fig. 12. In this system the hoist is driven by a direct-current shunt-wound motor, receiving power from the alternating-current supply-system through a synchronous or induction-motor-generator set. The hoist-motor is controlled by varying the voltage of the generator, which is separately excited, one generator being used for each motor.

The power- and current-curves for the balanced and unbalanced cycle are shown in Figs. 13 and 14, respectively. These curves show that the power consumed during acceleration is much smaller than for the induction hoist-motor, the efficiency then being approximately 80 per cent., and that a considerable part of the energy stored in the revolving parts of the hoist is returned to the supply-system as the hoist is brought to rest. On the other hand, the efficiency when the hoist-motor is running at full speed is lower than that for the induction hoist-

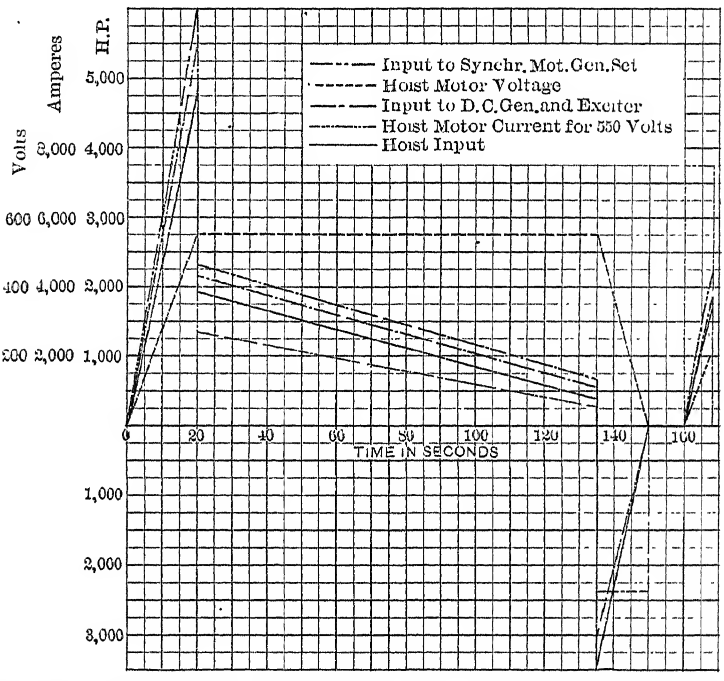


FIG. 13.—CURRENT- AND POWER-CURVES FOR A BALANCED CYCLE OF THE SYSTEM SHOWN IN FIG. 12.

motor, being approximately 82 per cent., and the losses of the motor-generator set when running light must be supplied during the time when the hoist is at rest. In view of the fact that a mine-hoist is idle 50 per cent. or more of the time under ordinary conditions, this is an item in the total power-consumption which cannot be neglected. It follows, from what has been stated, that the advantage of the direct-current hoist-motor over the induction hoist-motor in the efficiency through the complete cycle is greatest for short lifts, in which case the

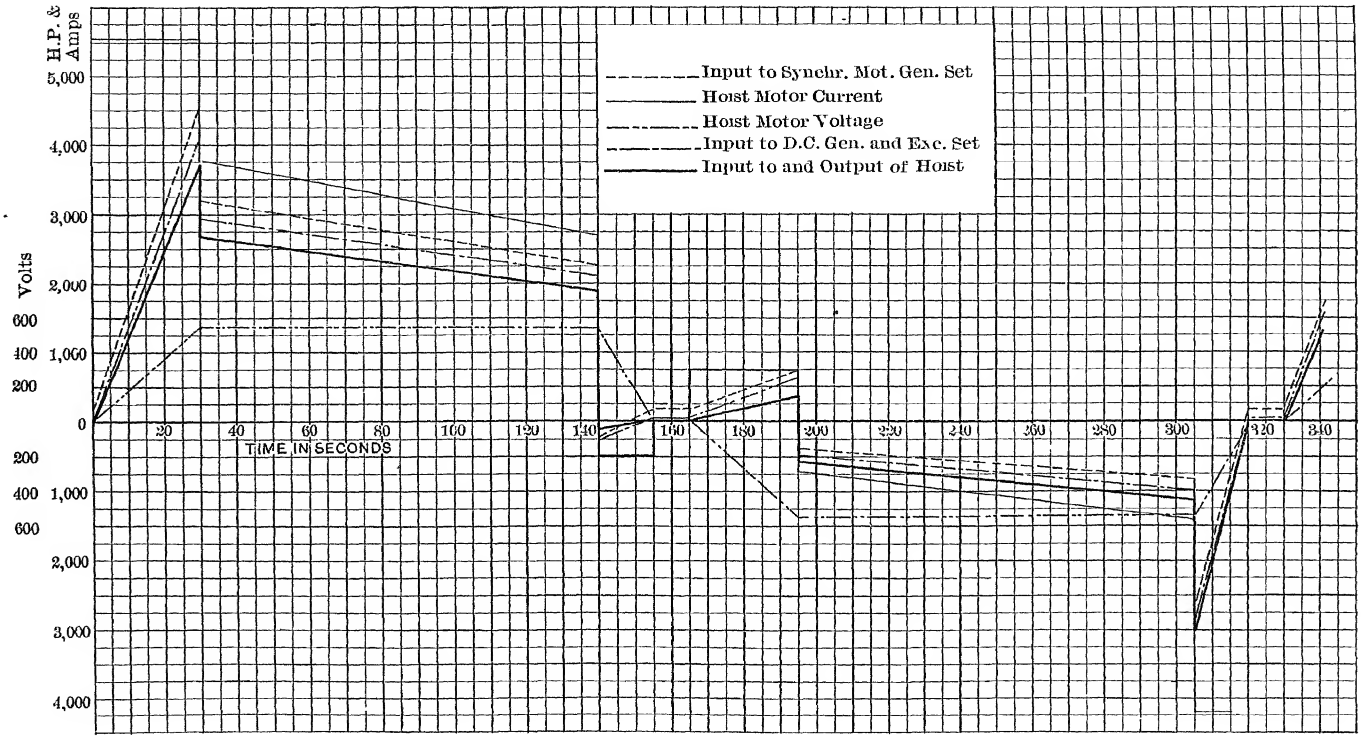


FIG. 14.—CURRENT- AND POWER-CURVES FOR AN UNBALANCED CYCLE OF THE SYSTEM SHOWN IN FIG. 12.

period of acceleration is a large percentage of the total cycle, and the time during which the hoist is idle is a minimum.

By reference to the curves of Fig. 14, it will be seen that approximately 30 per cent. of the power consumed in hoisting the ore unbalanced is returned to the system when the skip is lowered.

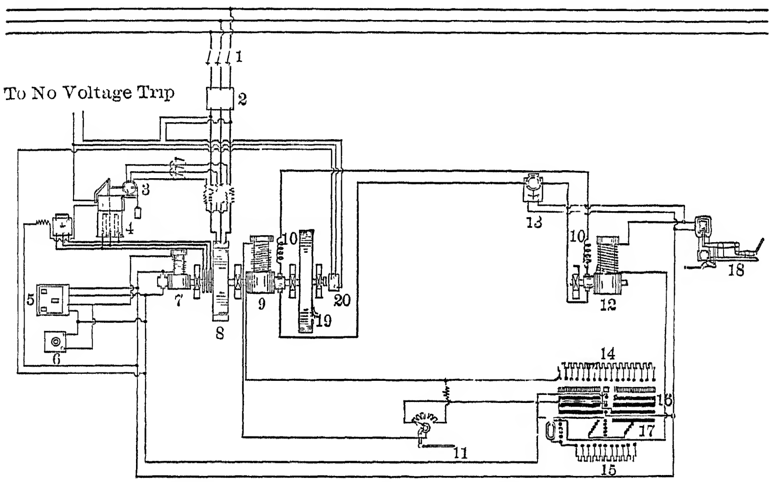
No definite rule can be laid down by which a choice can be made between the systems, each having advantages and disadvantages peculiar to itself which have a more or less important bearing on the choice, depending upon the special conditions of the individual problem. The first system has the advantages of low first-cost and simplicity, but is often at a disadvantage in respect to efficiency. On the other hand, the higher efficiency of the second system is frequently more than off-set by its increased first-cost and its greater cost of maintenance.

Both systems are open to the objection that the power drawn from the supply-system fluctuates between very wide limits during each cycle, generally reaching a maximum during acceleration, becoming negative during retardation for the second system, zero, or practically so, at the end of the cycle, and negative when lowering unbalanced for both systems. The effect of this wide fluctuation in the load during each cycle is to impair seriously the voltage-regulation of the supply-system unless its capacity is large as compared with the fluctuations, or unless the number of hoists driven from the same system is sufficient to produce a fairly-uniform load, which is seldom the case for a mine power-system. Also, if power is purchased, the price is usually made up of two components: one based on the total kilowatt-hours consumed, and the other on the maximum demand.

It therefore becomes necessary in most cases to provide some means whereby power may be taken from the supply-system and stored during the portion of the cycle when the demand for power is less than the average, and returned when the demand exceeds the average.

Fig. 15 shows such a system, the third, in which advantage is taken of the low first-cost and efficiency of the fly-wheel as a means for storing and returning large quantities of power for short intervals. This system is similar to the second, except

for the addition of a fly-wheel to the induction motor-generator set, and an automatic regulator for varying its speed. In its most common form, this regulator consists of a water-rheostat connected in series with the induction-motor armature. The resistance is varied by means of movable electrodes suspended from an arm mounted on the shaft of an induction-motor, which is connected in series, either directly or through series transformers, with the induction-motor of the fly-wheel set. The regulator-motor is so connected that its torque opposes the weight of the electrodes, which are partly counterbalanced to



1, 2. Line-switches. 3, 4. Slip-regulator. 5, 6. Tirrill regulator and rheostat. 7, 8, 9, 19. Fly-wheel motor-generator. 10. Commutating-pole and compensating field winding. 11. Safety-device. 12. Hoist-motor. 13. Circuit-breaker. 14, 15, 16, 17. Rheostat and controller of generator and hoist-motor. 18. Automatic brake solenoid. 20. Speed-limit device.

FIG. 15.—FLY-WHEEL MOTOR-GENERATOR SYSTEM.

reduce the size of the regulator-motor to a minimum, and permit of an adjustment of the regulator for different values of line-current. When the line-current exceeds the value for which the regulator is adjusted, the torque of the motor overbalances the weight of the electrodes, lifting them and inserting resistance in the armature circuit of the induction-motor. This causes it to slow down, and allows the fly-wheel to assist in driving the generator during the peak-loads. The sensitiveness of this regulator varies with the line-current, but within the range of ordinary operation the line-current may readily

be held within 5 per cent. either side of the mean, provided the hoisting is done at a uniform rate and the regulator is adjusted for the average load. In actual practice the regulator must be set for the maximum condition, so that under ordinary conditions of hoisting the current is limited rather than maintained constant.

The weight of the fly-wheel is determined from the hoist-diagram by correcting it for the losses in the hoist-motor and generator. Let Fig. 16 be such a diagram, and let a, b represent the average for the cycle. Then the energy represented by the portion of the diagram above the average-line

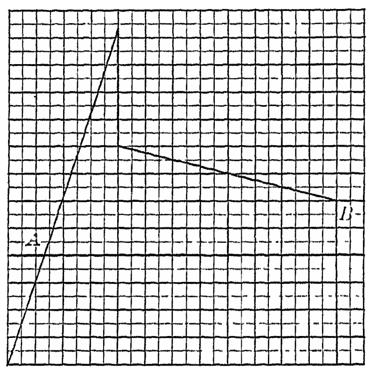


FIG. 16.—HOIST-DIAGRAM SHOWING AVERAGE FOR THE CYCLE.

must be delivered by the fly-wheel during the peak-load. The weight of the fly-wheel may be obtained from the equation

$$W = \frac{2\,g\,E}{Y_0^2 - Y_1^2},$$

where W = effective weight of wheel; Y_0 = the velocity of the wheel at a; Y_1 = the velocity of the wheel at b; and E = the energy in foot-pounds to be delivered by the wheel. It is the usual practice to make Y_0 approximately 300 ft. per second, in which case the actual weight of the wheel is approximately equal to 1.33 W.

A convenient method of obtaining the effective weight of a wheel is shown in Fig. 17. Having decided upon the radius of gyration in feet, and the speed in revolutions per minute, the weight of a wheel required to deliver any number of horse-

power-seconds from 0 to 100,000 with any change of speed from 10 per cent. to 70 per cent. may be obtained by the use of this diagram. To find the weight of a wheel, begin with the

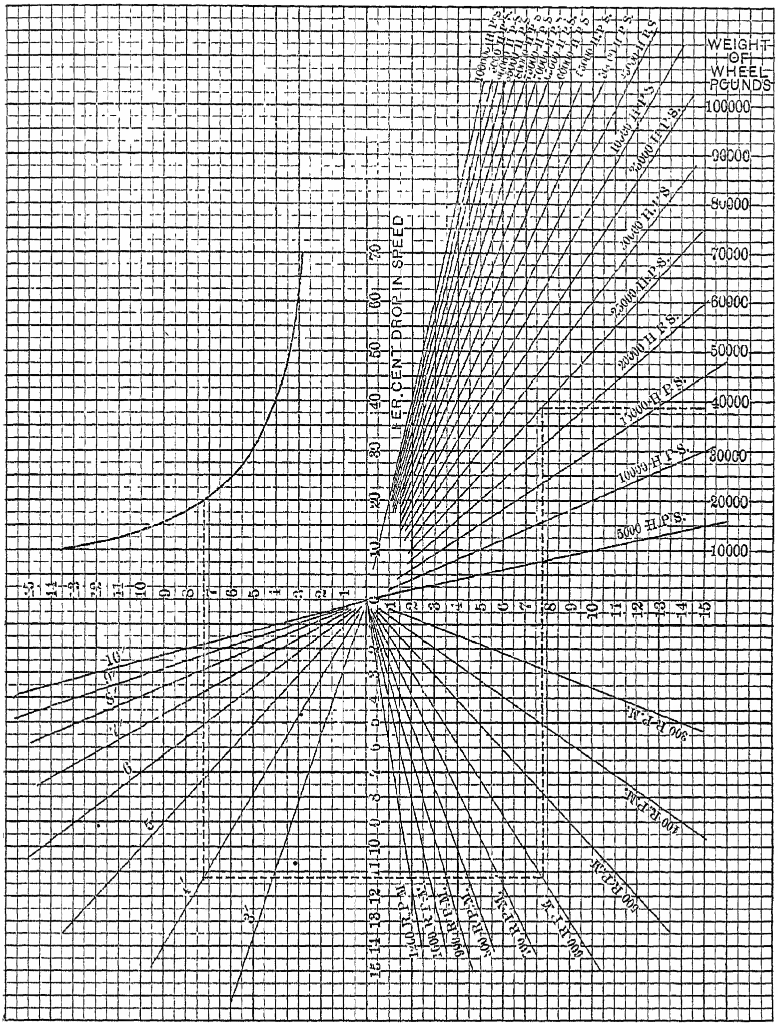


FIG. 17.—DIAGRAM FOR OBTAINING THE EFFECTIVE WEIGHT OF A FLY-WHEEL.

curve in the upper right-hand corner of the diagram, and follow the line corresponding to the per cent. change in speed until it intersects the curve, and then to the left until it intersects the line corresponding to the radius of gyration of the wheel, and

so on as indicated by the dotted line, which assumes a wheel having a radius of gyration of 4 ft., running at 600 rev. per min. and delivering 25,000 h-p-sec. with a 20-per cent. drop in speed. The effective weight of the wheel is approximately 39,000 lb. From the shape of the curve in the upper right-hand corner, it follows that the weight of the wheel increases very rapidly for drops in speed less than 15 per cent., and that

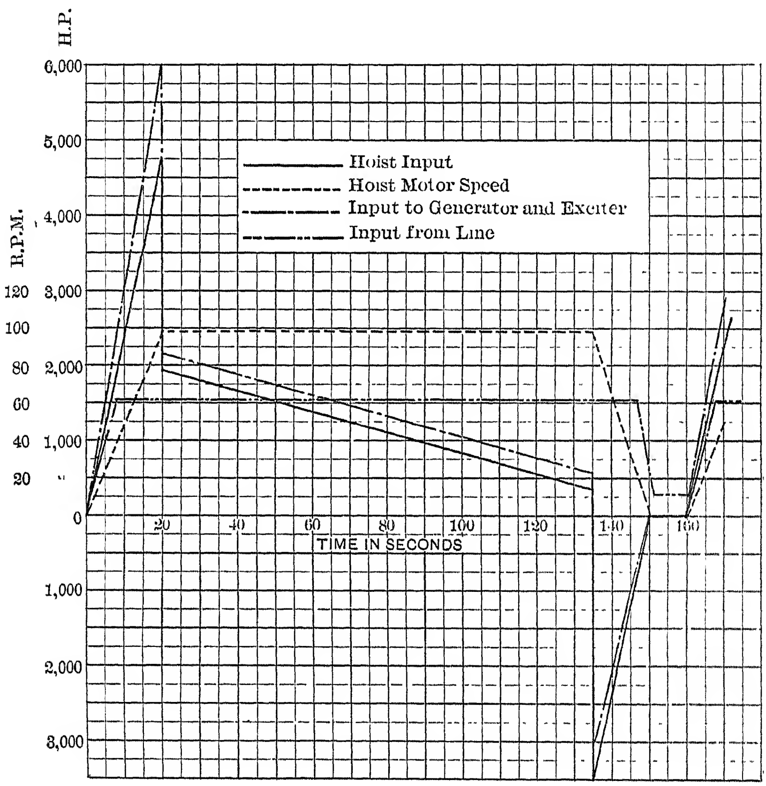


FIG. 18.—CURRENT- AND POWER-CURVES OF A BALANCED CYCLE OF THE SYSTEM SHOWN IN FIG. 15.

little is gained by increasing the drop beyond 35 per cent. On the other hand, the cost of the motor and generator decreases, and the efficiency of the induction-motor increases, and therefore the power consumed per cycle decreases, as the drop in speed decreases. The usual practice is to allow approximately 15 per cent. drop in speed for balanced operation; but as the fly-wheel must take care of the unbalanced cycle without re-

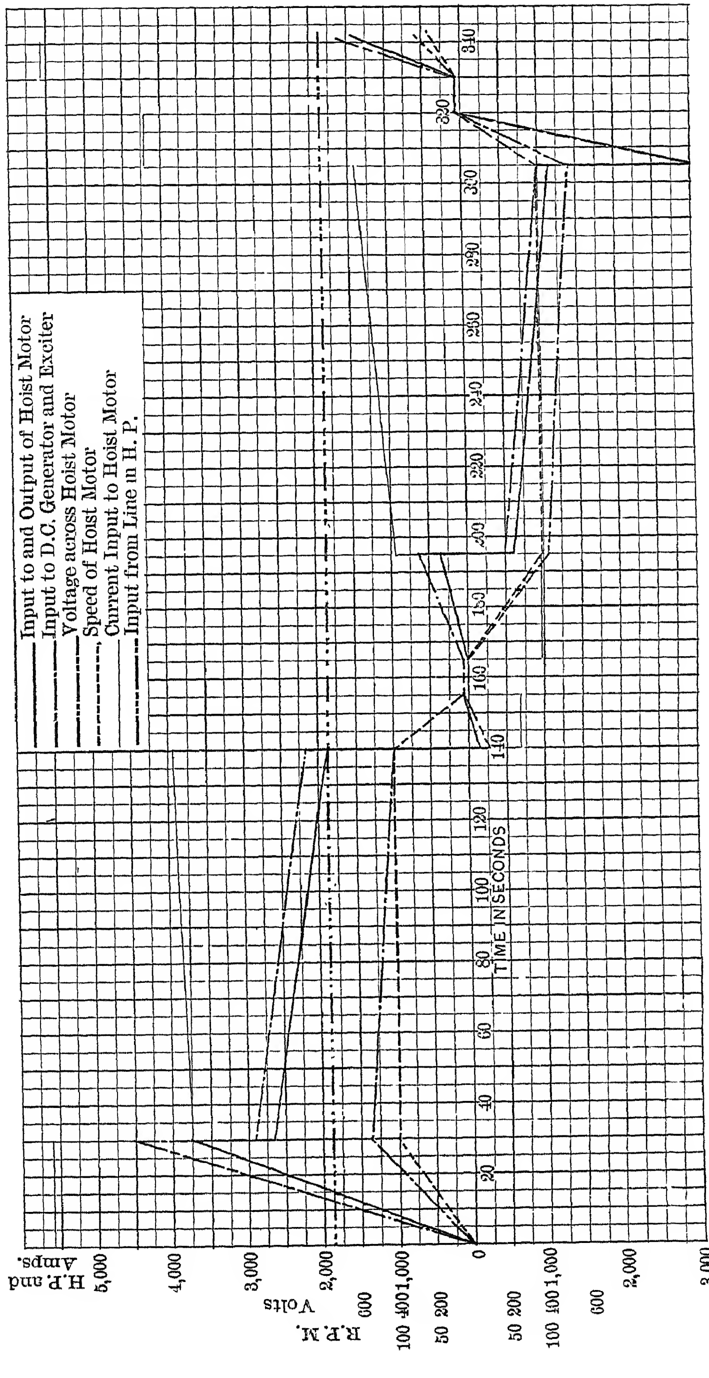
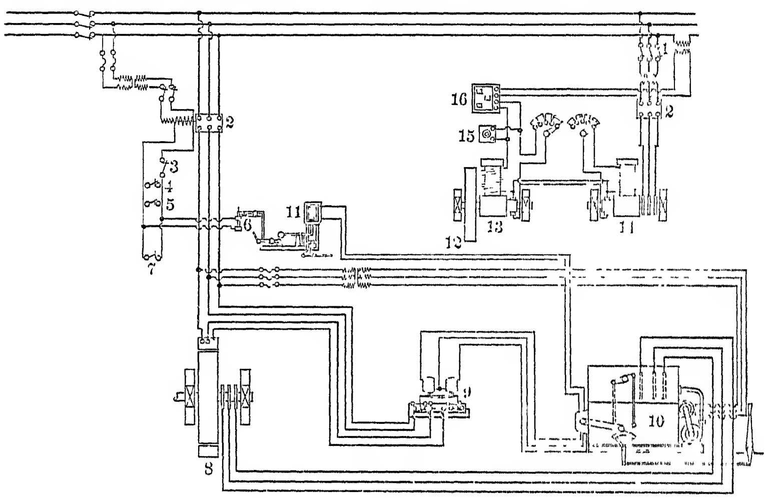


FIG. 19.—CURRENT- AND POWER-CURVES OF AN UNBALANCED CYCLE OF THE SYSTEM SHOWN IN FIG. 15.

ducing the speed of the generator so low as seriously to affect its commutation, it is necessary to vary this value considerably in special cases.

The power- and current-curves for this system are shown in Figs. 18 and 19. The drop in the power-curve of the balanced cycle during the period of rest is due to the fact that the regulator is set for the maximum condition, which in this case is hoisting unbalanced, this setting being above that required for the balanced cycle.

The fourth system is used when, for the purpose of meeting some peculiar condition, it is advisable to drive the hoist by an



1, 2. Line-switches. 3, 4, 5, 6, 7. Safety-switches for preventing overwinding and stopping the hoist. 8. Hoist-motor. 9. Reversing-switch. 10. Liquid controller. 11. Automatic brake. 12, 13. Fly-wheel dynamo. 14. Rotary converter. 15, 16. Tirrill regulator and rheostat.

FIG. 20.—CONVERTER-EQUALIZER SYSTEM.

induction-motor and at the same time eliminate the peaks from the station-load. The adoption of this system is warranted when the hoist is located underground at such a distance from the surface that it becomes necessary to transmit power to it by alternating current, and when the shaft is not large enough to allow the fly-wheel of the motor-generator set to be taken underground.

Fig. 20 shows this system, which, it will be noted, is the first system, to which has been added a converter equalizer, consisting of a rotary converter connected on the alternating-current

side to the supply-system, and on the direct-current side to a motor driving a large fly-wheel. The field of the direct-current motor is controlled by a regulator actuated by the line-current. When the power taken by the hoist-motor drops below the average, the field of the motor is automatically reduced, and the fly-wheel is speeded up, the power being taken from the supply-system. When the hoist-motor load exceeds the average the operation is reversed, the fly-wheel slowing down and returning power to the system.

Fig. 11 gives the current- and power-curves, which are those of the first system, to which has been added the input-curve with converter equalizer. The efficiency of this system is generally slightly lower, and the weight of the fly-wheel is slightly greater, than for the direct-current motor and the fly-wheel motor-generator set. It has the advantage, however, over the third system in that the operation of the hoist-motor is not dependent on the operation of a converter equalizer. Consequently, in the event of the failure of the latter, hoisting may be continued, providing, of course, that the capacity of the power-system is sufficient to take the load, which would be the case if the equalizer were used simply to reduce the power bill.

Either the third or the fourth system may be used where the supply-system is direct current, by substituting a direct-current motor for the induction-motor of the fly-wheel motor-generator set in the third system, and omitting the synchronous converter of the fly-wheel converter system in the fourth.

Among other systems of electric hoisting which might be mentioned is the Creplet system, in which a booster driving a heavy fly-wheel is connected in series with the hoist-motor and the supply-system. The booster is wound for the same potential as the supply-system, and the hoist-motor for twice this potential. When the hoist is idle, the hoist-motor armature is short-circuited, the booster is thus connected across the supply-system, and the speed of the fly-wheel is at its maximum. To accelerate the hoist, the short-circuit is opened, and the potential of the booster is generally reduced to zero, reversed, and brought up to full potential in the opposite direction, the power stored in the fly-wheel during the idle period being returned.

It has been proposed, and at least two installations embodying the idea are now in process of construction, to substitute

compressed air for steam. The present hoist-engines would be used, with slight modification of their valves to accommodate them to compressed air, the compressors for supplying the air to be driven by electric motors. It is impossible, however, to gather sufficient details regarding the system to predict the results which will be obtained.

A typical mine-hoist log is given in Table I., which is the condensed log for 24 hr., taken at a mine under actual conditions.

TABLE I.—*Mine-Hoist Log.*

Time in Minutes and Seconds.

Interval.	Hoisting Ore	Hoisting Men.	Hoisting Waste.	Other Hoisting.	Shifting.	Rest.
7–8 A.M.	.. ...	10–2	6–37	6–28	7–39	29–14
8–9	24–5	......	3–0	2–47	6–44	23–24
9–10	14–2	3–55	5–0	6–12	15–15	15–56
10–11	30–25	.....	12–0	1–30	8–35	7–30
11–12 M.	57–55	. ....	......	1–5	1–0	. ....
12–1 P.M.	3–35	1–0	17–5	2–55	4–29	30–56
1–2	51–10	1–50	......	1–45	4–0	1–15
2–3	20–10	......	......	2–40	12–5	25–5
3–4	44–0	......	......	0–51	5–25	9–41
4–5	......	7–38	......	1–25	0–38	50–19
5–6	......	......	......	......	0–35	59–25
6–7	......	13–8	......	1–23	0–27	45–2
7–8	21–27	......	16–42	1–54	2–1	17–56
8–9	51–7	1–8	......	4–57	1–59	0–49
9–10	54–17	......	......	1–30	1–30	2–43
10–11	48–22	1–42	......	......	2–53	7–03
11–12 M.M.	......	......	8–39	4–16	0–40	46–25
12–1 A.M.	34–51	1–52	......	.....	0–30	22–47
1–2	53–14	......	......	2–49	......	3–57
2–3	50–25	......	......	1–10	......	8–25
3–4	......	7–50	......	25–43	1–0	25–27
4–5	......	......	......	55–47	0–34	3–39
5–6	......	.....	......	5–14	......	54–46
6–7	......	......	......	3–09	......	56–51

Table II. gives a condensed summary of this log, and also that of another mine under actual operating-conditions.

TABLE II.—*Summary of Table I.*

	Mine A.			Mine B.	
	Minutes.	Seconds.	Per Cent.	Approximate Minutes.	Approximate Per Cent.
Hoisting ore and waste...	628	8	43.7	428	30
Other hoisting..............	185	15	12.9	171	12
Shifting......................	77	59	5.4	120	8
Rest..........................	548	38	38	721	50

The estimated distribution of power consumed for hoisting is given in Table III. Attention is called to the close agreement between the estimates for the power consumed in hoisting ore and waste, which, in view of the fact that the estimates were made entirely independent of each other, should add considerable weight to the figures. One estimate is based on figures obtained by indicating the hoisting-engine, and the other forms the basis for the distribution of the costs of hoisting.

TABLE III.—*Distribution of Power.*

	Distribution of Power in Per Cent. of Total.	
	Mine A.	Mine B.
Hoisting ore and waste	55	51
Other hoisting	28	23
Shifting	17	26

The curves in Fig. 21 give the total tons hoisted, and the total kilowatt-hours consumed (per day of 24 hr.), the kilowatt-hours consumed per ton- (2,000 lb.) foot, and the load-factor for each of the four systems when hoisting 10,000 lb. per trip in balance from vertical depths, varying from 400 to 2,600 ft., by a reel-hoist. Fig. 22 gives similar curves for hoisting 20,000 lb. of ore per trip in balance, from depths varying from 3,000 to 8,000 ft., by a cylindrical-drum hoist, the shaft making an angle of approximately 38° with the horizontal, and the depths being measured on the incline.

Table IV. gives the power-consumption for the various systems for hoisting 6,000 lb. in balance from one level in a compound shaft, the hoist load-diagram for which is given in Fig. 23. The shaft drops vertically 800 ft. and then continues for 1,320 ft. on an incline of 38° with the horizontal.

TABLE IV.—*Power-Consumption for Various Systems.*

	Power for Hoisting.	Power for Hoisting.		Total Tons Hoisted.	Load-Factor.
	Kw-hr.	Kw-hr.	Ton-ft.		
First system	3,900	0.00307[a]	0.00234[b]	786	25
Second system	3,450	0.00272	0.00206	786	19.2
Third system	4,019	0.00315	0.00240	786	62
Fourth system	4,910	0.00386	0.00295	786	63.6

[a] Values are based on the total vertical lift, 1,613 feet.
[b] Values are based on the total distance lift, 2,120 feet.

The values given in these curves and in Table IV. include the power consumed in "other hoisting" and "shifting," on

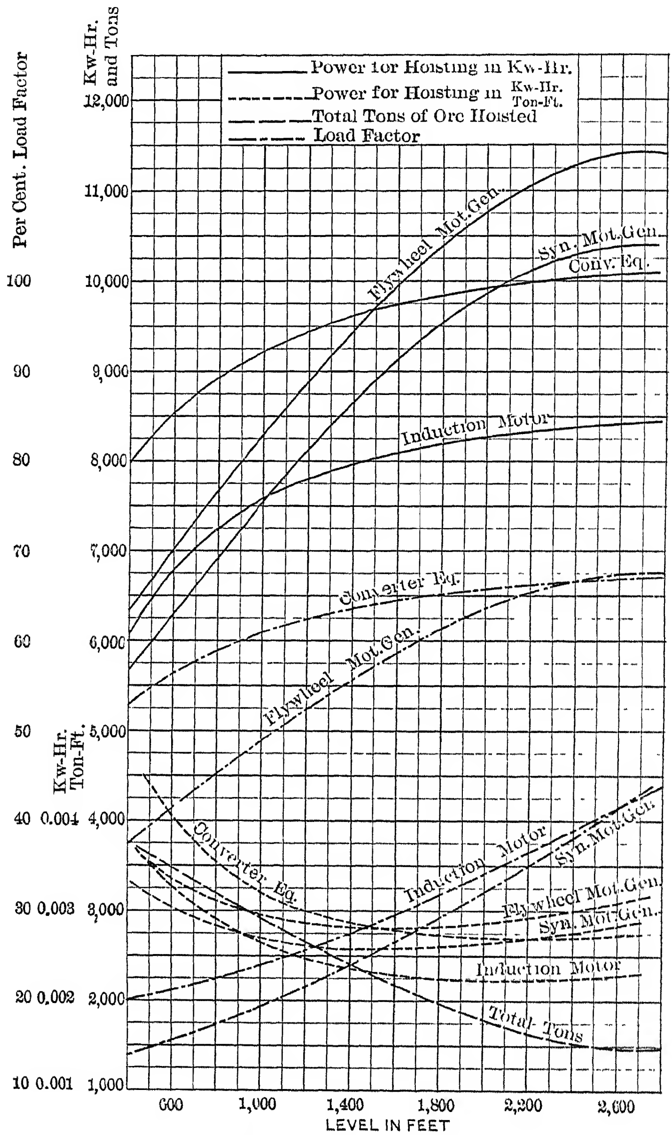


FIG. 21.—COMPARISON OF POWER-CURVES OF THE FOUR SYSTEMS, USING A REEL HOIST.

the assumption that the power consumed in hoisting ore is 53 per cent. of the total power. Also, in the values for the

synchronous motor-generator set (second system), no credit is allowed for the power returned during retardation.

The curves and Table IV. show clearly the effect of the shape of the hoist on the power consumed by the various systems. At the upper levels the period of acceleration is a large part of the total cycle, and we find the power consumed by the induction-motor hoist to be greater than that for the hoist driven by the direct-current motor, power for which is supplied by a synchronous or induction-motor-generator set. In the case of the cylindrical-drum hoist, for which the peak during acceleration is much greater than for the reel-hoist, the power consumed by the induction-motor is greater for all levels. Also, a similar relation exists in the power-curve when a converter equalizer or a fly-wheel motor-generator set is used.

If a cylindrical drum is used instead of a reel, the curves of Fig. 21 for the direct-current hoist-motors will remain practically the same, but those for the induction hoist-motor will be raised, crossing those of the synchronous motor-generator set towards the end of the curve; and if a cylindro-conical drum is substituted for the cylindrical drum of the larger hoist, the induction-motor curves of Fig. 22 will approach those for the direct-current hoist-motor.

The values for the different levels, shown by the curves, are based on the assumption that all the ore is taken from the corresponding level. Where ore is hoisted from several levels, the total power-consumption per day may readily be obtained from the power-consumption per ton-foot, or by the use of the curves of "Total Tons Hoisted" and "Power for Hoisting in Kw-hr."

While the curves given cover specific cases only, the examples chosen are typical, and by interpolating between the power consumed per ton-foot for the 8,000-ft. and that for the 2,500-ft. hoists, the power per ton-foot may be obtained for hoisting from any depth. Due correction is to be made for the inclination of the 8,000-ft. shaft by dividing the values given by 0.616 to obtain the equivalent values for a vertical shaft. From the total kilowatt-hours consumed per day and the load-factor, the cost of power for hoisting electrically can readily be obtained.

A comparison between steam and electric hoisting is given

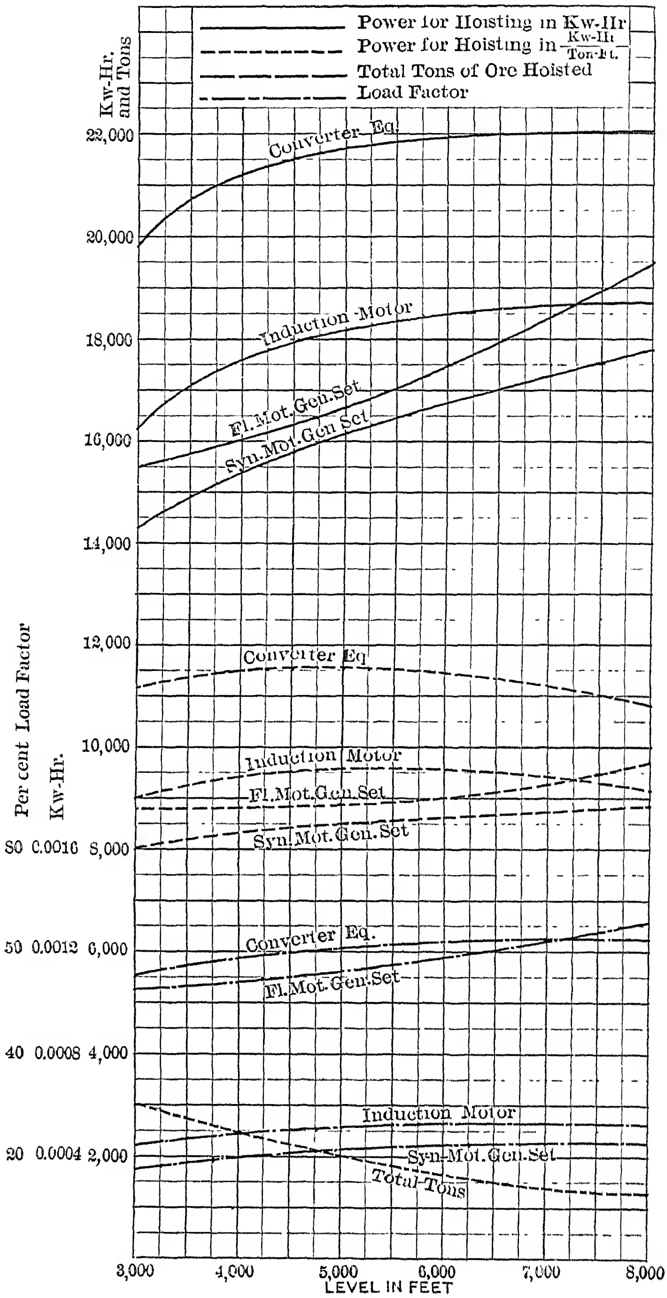


FIG. 22.—COMPARISON OF POWER-CURVES OF THE FOUR SYSTEMS, USING A CYLINDRICAL-DRUM HOIST.

in Table V., in which the coal-and-rock ratio for each is given for the 2,000-ft. and the 6,000-ft. levels respectively. In determining these values, it is assumed that the steam hoisting-engines are non-condensing, that the steam-consumption is 65 lb. and 55 lb. per indicated horse-power-hour respectively for the large and small hoists, and that power for the electric hoists is supplied from a modern steam-turbine station using units of 1,000 kw. each or larger for the small hoist, and 2,500 kw. or larger for the larger hoist.

In determining these ratios, 10 per cent. has been added to the total kilowatt-hours per day as given in the curves, to cover the losses in transmission.

TABLE V.—*Comparison of Steam and Electric Hoisting.*

Hoisting from 2,000-ft. level, small hoist:	Coal Burned. Tons per Day.	Ore Hoisted Tons per Day	$\frac{\text{Tons Ore.}}{\text{Tons Coal.}}$
Steam-hoist,	47.0	1,780	40
Electric hoist:			
First system,	13	1,780	137
Second system,	15	1,780	119
Third system,	16	1,780	110
Fourth system,	15	1,780	119
Hoisting from 6,000-ft. level, large hoist:			
Steam-hoist,	65.5	1,580	24
Electric hoist:			
First system,	23	1,580	69
Second system,	24	1,580	66
Third system,	25	1,580	63
Fourth system,	27	1,580	59

In addition to the saving in fuel, which may be realized by the use of electric hoists, there is a very material reduction in the labor, the cost of which is chargeable against the hoist. This may amount to the wages of one or two men in the boiler-house if power is developed by the mining company, or of the whole boiler-house force if power is purchased, and frequently the wages of one man in the hoist-house.

So many factors which vary between wide limits for different localities enter into the comparative costs of hoisting electrically and by steam, that each individual case must be treated by itself; but the following comparison will serve as an indication of the general result of a more detailed investigation.

Take, for example, the reel-hoist, the power-curves for which

are shown in Fig. 21, and assume that the average condition of hoisting is that represented by hoisting from the 2,000-ft. level; that good steaming-coal can be purchased for $3.50 per ton; that power can be purchased for the equivalent of 1.1 cents per kilowatt-hour, on a 50-per cent. load-factor, and, if steam driven, that the engine will be non-condensing.

In order to obtain a load-factor of 50 per cent., it will be necessary to install either the third or fourth system of electric

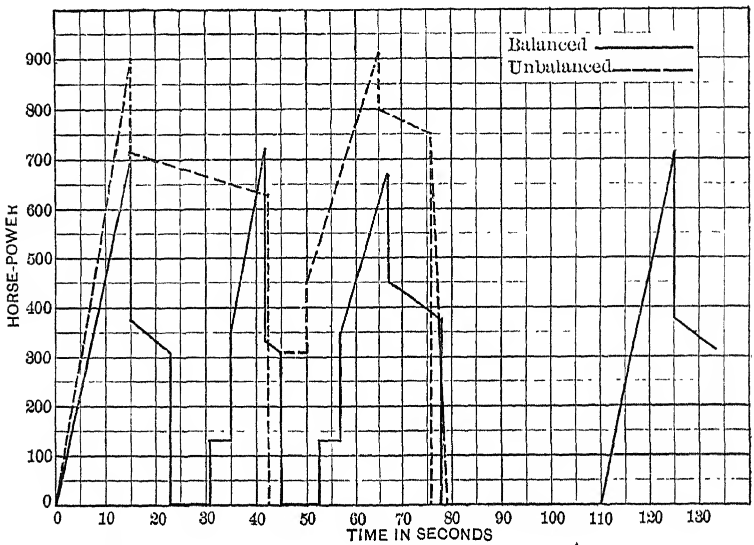


FIG. 23.—HOIST-LOAD DIAGRAM FOR 6,000-LB. LOAD IN A COMPOUND SHAFT.

hoisting, and, to be conservative, let it be assumed that the third is chosen.

Total cost of power per year for electric hoist at 1.1 cents per kilowatt-hour,	$35,300
Fixed charges on the excess cost of the electric over the steam hoist (approximately $25,000),	2,500
	$37,800
14,100 tons coal at $3.50 per ton,	$49,350
Boiler-house force (3 men at $3.25 per day),	2,925
One oiler,	900
	$53,175
	37,800
Approximate annual saving with electric hoist,	$15,375

As this saving of $15,375 is the result of an additional expenditure of $25,000 for the electric hoist, it is proper for a new installation to base the rate of interest equivalent to this saving on this additional first-cost, from which it follows that the interest realized on this investment is 61.5 per cent. If, on the other hand, it is a question of replacing an existing steam-hoist, the interest should be based on the total cost of the electric installation, in which case the very substantial rate of 30 per cent. will be realized.

The hoist, above all other parts of the mine-equipment, must be kept in commission at all times, and this fact must be borne in mind in installing an electric hoist. The transmission-lines must be carried on substantial poles over a well-cleared right-of-way, duplicate lines being installed where possible, and the lines being adequately protected against disturbances from lightning. If these precautions are properly taken, the electric can be made as thoroughly reliable as the steam-driven hoist.

V. Summary.

Summing up, the advantages of the electric hoist are: first, greater economy, resulting from the centralization of the development of power in a large central electric station, favorably located for the economical development of power, and from a reduction in the operating-force, from the increased life of the rope, and the greater life of the brakes; second, greater safety in operation; third, especially, adaptation for underground installations; and fourth, the fact that it permits of the utilization of water-power, which is frequently available in mining-districts.

VI. Bibliography.

ABBREVIATIONS.

Al. Ind.,	*Alliance Industrielle.*
Am. Elec'n.,	*American Electrician.*
Bull. As. Ing. El.,	*Bulletin de la Association des Ingenieurs Electriciens Sortis de l'Institut Electrotecnique Montefiore*, Liége.
Bull. Soc. belge d'Elec.,	*Bulletin de la Société belge d'Electriciens.*
Col. Guard.,	*Colliery Guardian.*
D. R. P.,	Deutsches Reich Patent.
Ding. Poly,	*Dingler's Polytechniches Journal.*
E. & M. Jour.,	*Engineering and Mining Journal.*

E. T. Zeit.,	*Electrotechnische Zeitschrift.*
Eclair. Elec.,	*L'Eclairage Electrique.*
El. Anzeig.,	*Elektrotechnischer Anzeiger.*
El. Bahn.,	*Elektrische Bahnen.*
El. Eng'g.,	*Electrical Engineering*, London
El. Engr.,	*Electrical Engineer.*
El. Jour.,	*Electric Journal.*
El. Kr. u. Bahn.,	*Elektrische Kraft-Betriebe und Bahnen.*
El. Rev., N. Y.,	*Electrical Review, New York.*
El. Rev., Lond ,	*Electrical Review, London.*
El. u. Masch.,	*Elektrotechnik und Maschinenbau.*
El. Wld.,	*Electrical World.*
Elec'n, Lond.,	*Electrician*, London.
Elec'n, Paris,	*Electricien*, Paris.
Eng'g Mag.,	*Engineering Magazine.*
Eng'g News,	*Engineering News.*
Eng'g.,	*Engineering*, London.
Engr.,	*Engineer*, London.
Glück.,	*Glückauf.*
Gen. Civ.,	*Genie Civil.*
Ind. Elec.,	*L'Industrie Electrique.*
I. T. Rev.,	*Iron Trade Review.*
I. & C. T. Rev.,	*Iron and Coal Trades Review.*
Jour. I. E. E.,	*Journal of the Institution of Electrical Engineers.*
Lum. Elec.,	*La Lumière Electrique.*
Min. & Min.,	*Mines and Minerals.*
Min. Jour.,	*Mining Journal.*
Min. Rep.,	*Mining Reporter.*
Mech. Engr.,	*Mechanical Engineer.*
Oes. Zeit. f. B. u. H. W.,	*Oesterreichische Zeitschrift fur Berg- und Hüttenwesen.*
Prac. Engr.,	*Practical Engineer.*
Proc. I. C. E.,	*Proceedings of the Institution of Civil Engineers.*
Proc. I. Mech. E.,	*Proceedings of the Institution of Mechanical Engineers.*
Rev. de Méc.,	*Revue de Mécanique.*
Rev. Elec.,	*Revue Electrique.*
Sch. of M. Quart.,	*School of Mines Quarterly.*
Schw. Elek. Zeit.,	*Schweizerische Elektrotechnische Zeitschrift.*
Sci. Am.,	*Scientific American.*
Trans. A. I. E. E.,	*Transactions of the American Institute of Electrical Engineers.*
Trans. A. I. M. E.,	*Transactions of the American Institute of Mining Engineers.*
Trans. Inst. M. & M.,	*Transactions of the Institution of Mining and Metallurgy.*
West. Elec'n.,	*Western Electrician.*
Zeit. d. oes. Ing. u. Ark. Ver.,	*Zeitschrift des oesterreichische Ingenieure und Architekten Vereins.*
Zeit. f. Elek.,	*Zeitschrift für Elektrotechnik.*
Zeit. f. Elek. u. Masch.,	*Zeitschrift für Elektrotechnik und Maschinenbau.*
Zeit. V. d. I [illegible]	*Zeitschrift des* [illegible]

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Tests of an Ilgner Electric Hoist.*

BY R. R. SEEBER, WINONA, MICH.

(Pittsburg Meeting, March, 1910.)

IN the copper-mining district of northern Michigan a fair-sized mine usually operates two or more shafts along the strike of the lode, these shafts being usually at least 1,000 ft. apart. The tonnage to be hoisted from each shaft is large—from 13,000 to 25,000 tons per month. The hoisting-engines used are generally steam-driven, non-condensing, duplex engines with Corliss valves, supplied by steam from separate boiler-plants where the distance between shafts is great.

In an attempt to centralize power-plants and secure central-station economies, the Winona Copper Co. installed an electric hoist of the Ilgner type, with Ward Leonard control. A brief description of this plant has already been published.[1] The only point of variation from the simple Ilgner set lies in the connection of two generators to a single induction-motor by flexible couplings. These generators serve motors which operate hoisting-drums at separate shafts, viz., No. 4 shaft, Winona, and No. 1 shaft, King Philip. The hoist for No. 4 shaft is about 1,700 ft. from the power-plant, and in this building the motor-generator set is placed. No. 1 King Philip hoist is 1,500 ft. from the motor-generator set. The equipment of the motor-generator set is as follows:

In the middle of the set is a 12-pole, 450-h-p., 600-rev. per min., 2,080-volt, three-phase, 60-cycle, variable-speed induction-motor, connected on each side to a 20-ton fly-wheel which is 10 ft. in diameter, and to a 6-pole, 170-kw, 575-volt, interpole, direct-current generator. On each side of the shaft is placed a separate exciter, which delivers current at 125 volts. The lubrication of the four main bearings that support the fly-wheels

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* Presented also, by mutual agreement, at the meeting of the American Institute of Electrical Engineers, New York, N. Y., Mar. 11, 1910.

[1] *Engineering and Mining Journal*, vol. lxxxviii., No. 3, p. 110 (July 17, 1909).

and direct-current generators is supplied by two sets of oil-pumps, belt-driven from the shaft.

The hoist-motors are 6-pole, 200-h-p., 430-rev. per min. 550-volt motors, and are shunt-wound. The fields, both of this motor and the direct-current generators, are excited by current from the exciter on the motor-generator set, which is kept at a constant voltage of 125 by voltage-regulators. The current in the hoist-motors is varied and reversed through controllers of the usual railway type, placed in the field-circuit of the generators. The first point on the controller operates a contactor which closes the main circuit from the generator to the motor.

The alternating current for operating the motor-generator passes through a regulator which automatically varies the resistance in the rotor of the motor-generator set in proportion to the demand of the motor for current.

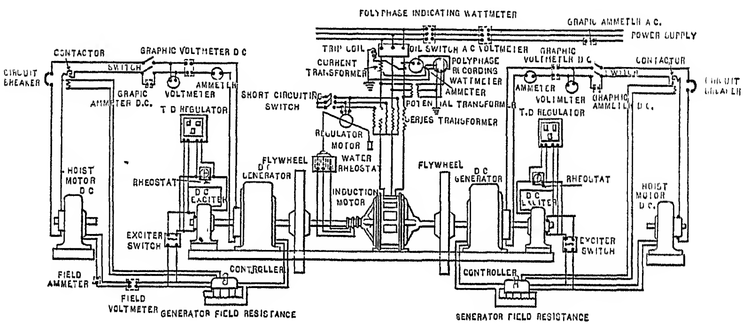

Fig. 1.—Diagram of Connections of Motor-Generator Set.

During February, 1910, detailed tests of this set were made to determine the performance under varying conditions, and to procure coal-to-rock ratios to compare with ratios for steam-hoists under similar conditions.

On the switchboard controlling the set, the following instruments were already installed:

Alternating-current indicating ammeter.
Recording wattmeter.
Direct-current ammeter and voltmeter for each direct-current generator.

In addition, for the purposes of this test, the following instruments were installed, as shown on the diagram, Fig. 1:

Two 600-0-600 direct-current graphic voltmeters, one in each generator circuit.
Two 800-0-80 direct-current graphic ammeters, one in each generator circuit.
One 125-ampere alternating-current graphic ammeter.
One 125-volt direct-current indicating voltmeter.
One 100-ampere direct-current indicating ammeter
One indicating wattmeter with necessary current-transformers.
One 600-rev. per min. tachometer.

No. 4 shaft Winona is served by the hoist-drum in the same building as the motor-generator set. This is an old duplex steam-hoist, with the cranks disconnected. The motor drives the old crank-shaft through gearing. The drum is 7 ft. in diameter and holds 1,500 ft. of rope. The rope is $1\frac{1}{8}$ in. in diameter and weighs 2 lb. per foot. No. 4 shaft is double-compartment and the skips work in balance. Because at present very little hoisting is being done in this shaft, only the south skip is used for loads and the north skip carries 2,100 lb. of rock to lessen the starting-moment. At the time of the test the north skip had filled with ice, so that the total counterbalancing weight was 3,300 lb. This caused more power to be used for the lowering of the south skip than for hoisting it. The loaded skip weighed 5,400 lb. Of this amount, 3,300 lb. is counterbalanced, leaving only 2,100 lb. of rock and the hoisting-rope as the net load.

No. 1 shaft King Philip is operated by an old geared steam-hoist, with the cranks disconnected and the crank-shaft geared to the motor in the same manner as at No. 4 Winona. The drum is 5 ft. in diameter and holds 1,200 ft. of 1-in. rope, weighing 1.58 lb. per foot. The shaft is built with two compartments, but only one skip is running, leaving the hoist unbalanced.

The skips in both shafts are similar, each weighing 2,900 lb. and holding 2.7 tons of rock when full. The test was conducted under normal working-conditions and the loads varied. During the time of test, a man was stationed at the brace of each shaft to observe the fullness of the skips. The average distance of the rock from the top of the skip was estimated for each load and a correction was applied.

The work at present being done at these shafts is comparatively small, because only development-work is under way. Since all reliable information that I had concerning the operation of steam-hoists was at a hoisting-rate of from 12,000 to 15,000 tons per month, we arranged to hoist from each shaft

for an hour at near this rate. During all this time the curve-drawing instruments were in operation and gave accurate records of each operation. Readings of the alternating-current wattmeter, voltmeter, and ammeter were taken every 5 sec. The starting- and stopping-time of each of the hoist movements was taken, and the nature of the load recorded. The speed of the motor-generator set was taken at regular intervals.

Tables I. and II. show the levels from which rock was hoisted during this 1-hr. test, also the total number of skips, total rock hoisted from each level, the average load, the distance from the level to the dump on the incline of the shaft, 70°, and the product of load and distances; also totals and averages. This rate of hoisting corresponds to about 19,000 tons per month, or rather more than the rate of the steam-hoists with which it is compared.

TABLE I.—*Operations at the Winona Shaft.*

Level.	Skips	Load. Total.	Load. Average.	Distance.	Load-Distance.
		Tons.	Tons.	Feet.	Feet.
9th	5	11.9	2.4	918	10,924
10th	4	10.2	2.5	1,018	10,384
11th	4	10.2	2.5	1,118	11,403
12th	3	7.7	2.6	1,218	9,379
	16	40	2.5	Average 1,052	42,090 At 600 ft. equivalent to 70.15 tons.

TABLE II.—*Operations at the King Philip Shaft.*

Level.	Skips.	Load. Total.	Load. Average.	Distance.	Load-Distance.
		Tons.	Tons.	Feet.	Feet.
5th	1	2.0	2.0	425	850
6th	6	13.2	2.0	524	6,917
7th	...	......	......	623	....
8th	5	10.2	2.04	729	7,436
9th	3	4.95	1.65	832	4,119
10th	4	9.45	2.36	932	8,807
11th	...	......	......	1,034	....
12th	3	7.4	2.5	1,135	8,399
	22	47.2	2.15	Average 774	36,528 At 600 ft equivalent to 60.88 tons.

Adding the total tons hoisted at Winona to the total tons hoisted at King Philip during the same period gives an equivalent of 131 tons hoisted from a depth of 600 ft. in a 70° incline. The total kilowatt-hours taken by the motor-generator set during this hour was 211. This output was determined by reading an indicating wattmeter every 5 sec. and taking the average. The power necessary to hoist 1 net ton of rock from a depth of 600 ft. on the incline was 1.61 kilowatt-hours.

Tests of the power-plant, under present light-load conditions, have shown that it requires about 4 lb. of coal to produce 1 kw-hr. delivered on the switch-board of the motor-generator set. The amount of coal required to hoist a ton from this distance is therefore 6.44 lb., giving a coal-to-rock ratio of 1 to 310:

Winona, . . .	70.15 tons.
King Philip, . .	60.88 tons.
	131.03 tons hoisted at 600 ft depth on 70°.

211 kw-hr. taken by set.
1.61 kw-hr. per ton.
6.44 lb. coal per ton at 4 lb. coal per kilowatt-hour.
Coal-to-rock ratio = 1 to 310.

For purposes of comparison with the steam-hoist conditions, I have tabulated the results in rock hoisted and kilowatt-hours taken by the motor-generator set for an average day:

16 hr.	211 kw. =	3,376 kw-hr.
4 hr.	80 kw. =	320 kw-hr.
1 hr.	55 kw. =	55 kw-hr.
	Total of	3,751 kw-hr.

3 hr. shut down.
Total of 24 hr.

Rock is hoisted during but 16 hr. of the time, making the total rock hoisted for both shafts 2,096 tons; or 1.79 kw-hr. are required to hoist 1 net ton of rock from a depth of 600 ft. on a 70° incline. This makes the coal-to-rock ratio for the above conditions 1 to 279.

The theoretical value of kilowatt-hours per ton hoisted, taken from Fig. 21 of the paper Electric Mine-Hoists, by Rushmore and Pauly,[2] is 1.7 kw-hr. Taken from the curves in Fig. 22 of

[2] This volume, pp. 58 to 108.

the paper referred to, for the 8,000-ft. hoist, this value is 1.55 kilowatt-hours.

At the D shaft of the Champion Copper Co., a 24- by 60-in. duplex Corliss engine operates a double conical hoist-drum. The shaft-inclination is about 70°, the skips working in balance. The boiler-plant supplies the rock-house engine in addition to supplying the hoist. For the first four days of November, 1905, the boiler-plant supplied the hoist alone. The coal-to-rock ratio for these four days, at 600 ft. average depth, was 1 to 154. Using a percentage of fuel burned for the rock hoisted, obtained from results of this four days' test (68.4 per cent. to hoist), the ratio for the remainder of November was 1 to 185, showing gain by better boiler-load. For December of the same year, on the same basis, the ratio was 1 to 166. The average load of rock was 2.1 tons, the total amount hoisted in a month being about 15,000 tons.

At the Winona mine, No. 3 shaft was operated in June, 1907, by a separately-fired boiler-plant. The hoist is a duplex geared hoist with a rolling valve. The shaft is inclined 70° from the horizontal and the skips were in balance. The average load of rock was about 2.1 tons. In June, the coal-to-rock ratio at 600 ft. depth, figured as before, was 1 to 124. In August and September, 1907, this hoist was supplied from a larger plant, the hoist being from 15 to 20 per cent. of the whole load. The distribution of coal gives a ratio of 175 to 200.

The following tabulation compares these ratios:

Electric hoist, two shafts, Winona ; coal-to-rock ratio 1 to 279.
Champion D hoist, steam (alone) ; coal-to-rock ratio 1 to 154.
Champion D hoist, with rock-house ; coal-to-rock ratio 1 to 185.
Winona No. 3 hoist, steam (alone) ; coal-to-rock ratio 1 to 124.
Winona No. 3 hoist, steam, (central plant) ; coal-to-rock ratio 1 to 175, to 1 to 200.

I feel satisfied from this data that, even with steam generated in a central plant, a coal-to-rock ratio of 1 to 200 is about all that can be expected from steam-hoists of this type, under the given conditions. This shows a coal-saving by the electric method of 25 per cent. over the steam method.

Fig. 2 shows the operation of the motor-generator set and both hoist-motors for a typical section of about 9 min. length, taken from the 1-hr. test. The curves here shown will serve

to describe most of the operation. The top curve shows the alternating-current input plotted from 5-sec. readings of the indicating wattmeter. The average for the 9 min. is 208.25 kw., or slightly below the average input for the hour.

The second curve shows the speed of the motor-generator

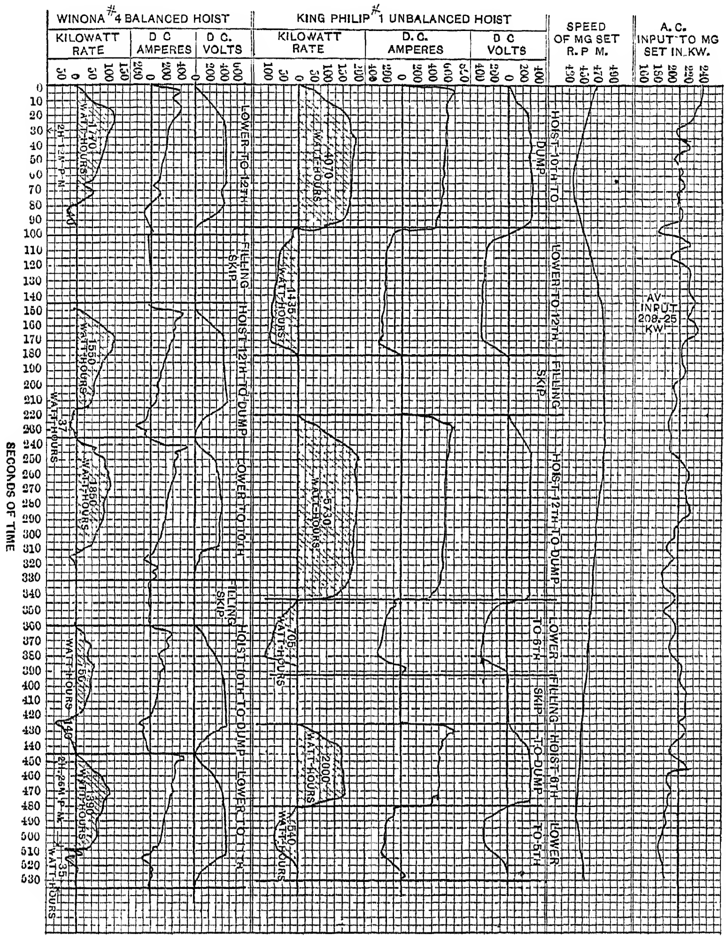

FIG. 2.—OPERATION OF THE MOTOR-GENERATOR SET.

set, the extreme variation being from 440 to 480 rev. per min. during this period.

The next three curves show the volt-, ampere-, and kilowatt-output of the direct-current generator driving the No. 1 King Philip motor.

The last three curves show the volt-, ampere-, and kilowatt-output of the direct-current generator driving the Winona No. 4 hoist for the same period.

Where electricity is used for hoisting, the main reason for using an Ilgner fly-wheel set is to take the peak-loads off the power-station. How well this is accomplished is shown by comparing the input-curve of the diagram with the two curves of direct-current output. Incidentally this method utilizes the energy of a descending unbalanced skip; the motor acting as a generator and restoring energy in the fly-wheel. For the King

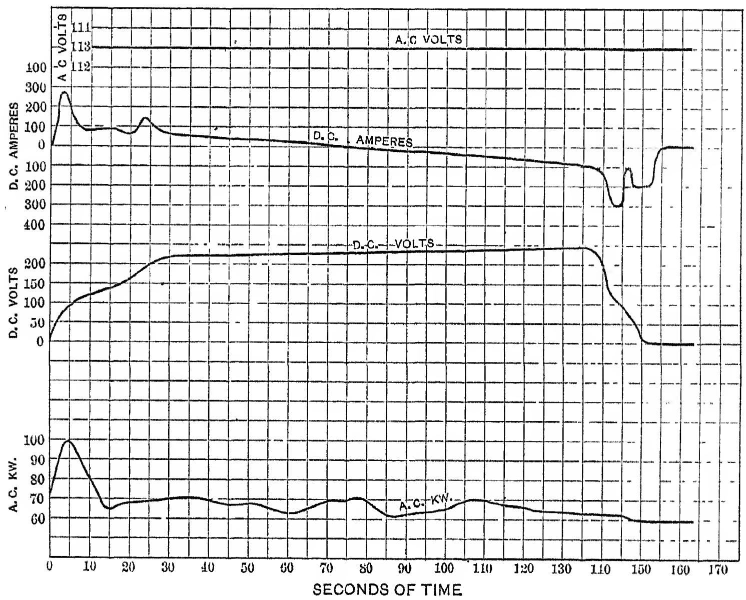

FIG. 3.—RESTORATION OF ENERGY.

Philip hoist, unbalanced, the amount thus restored is comparatively large, varying from 20 per cent. to 35 per cent. of the energy required to hoist the loaded skip from the corresponding level. This variation is due to variation in the loading of the skips. On the Winona hoist, balanced, the amount of energy thus restored is very small and occurs only when the motor is used as a brake at the close of the hoisting-period.

The restored energy for both cases is shown on the shaded areas below the line. An interesting example of this restoring

action is shown in Fig. 3 on Winona No. 4 hoist, where men are being hoisted on the south skip. Energy is required at starting, but the demand gradually decreases, and at about the 8th level the weight of the descending north skip counterbalances the ascending skip. During the remainder of the hoisting-period energy is being stored in the fly-wheel.

Where a double-compartment shaft can be used and the skip and rope can be counterbalanced by duplicates in the other compartment, the counterbalancing skip being ready for a paying load on reaching the bottom, it is obviously not economy of power to use any other system of restoring the energy of the descending skip. The efficiency of any other system will

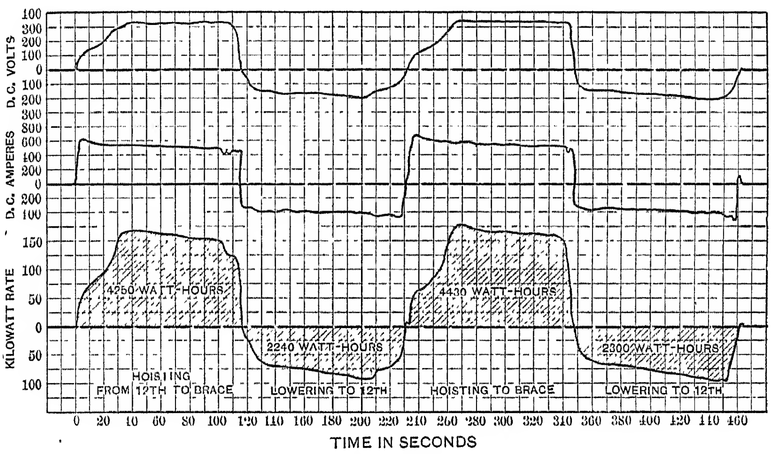

FIG. 4.—ENERGY EXPENDED IN HOISTING AND LOWERING UNBALANCED HOIST.

be very low compared to the efficiency of the counterbalancing skip, where the only losses are friction and air-resistance.

Fig. 4 shows the current-output of the King Philip generator when hoisting a full skip from the 12th level, and the energy restored to the set when lowering the same skip still full. This was done twice, as shown.

The watts used in hoisting the first time were 4,250; watts returned lowering were 2,243, showing an efficiency of restoration of 52.8 per cent. The watts used hoisting the second time were 4,430; the number of watts returned lowering the second time was 2,300, showing an efficiency of restoration of 51.9 per cent.

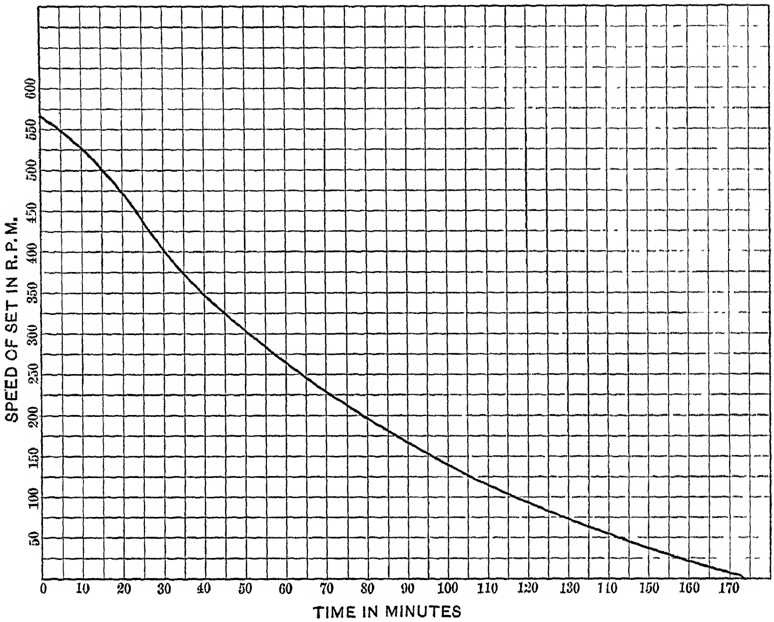

Fig. 5.—Retardation of Motor-Generator Set.

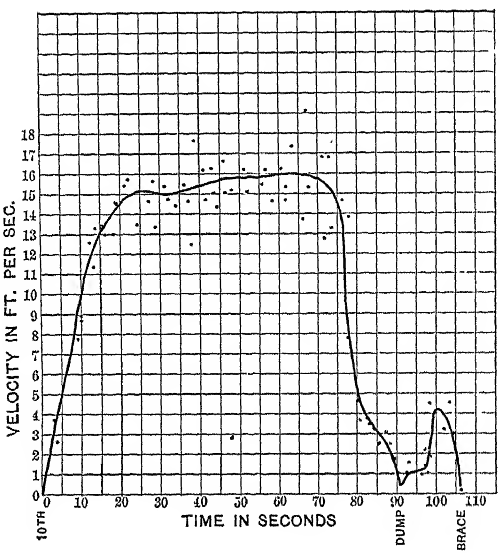

Fig. 6.—Velocities of Hoisting.

Fig. 5 shows the decreasing speed of the motor-generator set, after the power is shut off on Saturday nights. About 3 hr. are required for the set to come to rest.

Fig. 6 shows the acceleration of the skip and the rope-speed for the hoisting-period.

The greatest disadvantage of the Ilgner system is a constant loss in windage and bearing-friction of the motor-generator set itself. Observations have shown this loss for the set at this plant to be very nearly 55 kw. This loss can be reduced materially by casing the fly-wheels. When the work done by the hoist is large enough to make the proportion of this constant loss small, an Ilgner fly-wheel set adds a saving of fuel to the saving of labor, etc., due to the centralization of the power-plant.

The use of this system enables us to dispense with two boiler-plants and their firemen. It would have been necessary, also, to build larger boiler-plants, with railroads and coal-trestles to serve them. The hoists themselves handle the skips much better than the steam-hoists, starting from rest very smoothly. The large amount of repair-work necessary on a reciprocating engine is also eliminated.

The Girod Electric Furnace, and the French Works Using the Paul Girod Steel-Process.*

BY DR. WILHELM BORCHERS.†

(Pittsburg Meeting, March, 1910.)

In all special branches of the chemical and metallurgical industries, in which large electric furnaces became necessary for carrying out new processes or for the improvement of old ones, the development of the electric furnaces took place on much the same lines. The endeavors to make a success of a new idea brought forth many ingenious designs, which, however, almost without exception, furnished the proof that the safest and shortest way to practical success is the utmost simplicity of construction of the necessary apparatus.

Our knowledge of the qualities of materials for building electric furnaces and of the constituents of the furnace-charges, especially their electric conductivity and their chemical affinity towards one another at high temperatures, is still far from being complete, and from this fact we must conclude that the smaller the number of elements in the construction of a furnace, the fewer the number of unreliable factors both in the construction and in the operation of the electric furnace.

This conclusion deserves special attention when an electric furnace for making steel has to be chosen from the constantly-growing number of inventions, inasmuch as the chief constituent of steel, the element iron, is a substance which in its liquid phase has so great an affinity for those elements which are to be removed from the charge, that it is very difficult to comply with the following conditions: 1, to heat the refining-slag to the highest point of its chemical activity; 2, to prevent the iron from entering the slag, or, where this cannot be done, to make it enter so that it finally appears in the end-product

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† Professor of Metallurgy, Royal College of Technology, Aachen, Germany.

without material loss; 3, to select and arrange the materials from which the furnace is to be built, so that wherever contact with the furnace-charge cannot be avoided, these materials will not react upon the constituents of the charge in an injurious way.

About six years ago, in my review of the electrode furnaces up to that time,[1] I could say that a furnace simpler than the Héroult furnace could not be found, but to-day I have to correct this statement. At the present time the Girod furnace must be recognized as the simplest and safest, both in construction and operation, of all electrode furnaces.

As far as I am from changing my good opinion of Dr. Héroult's achievements in electro-metallurgy, I consider it my duty as professor of electro-metallurgy, and as an author of electro-metallurgical books and other scientific publications, to correct publicly opinions of former stages of scientific progress, which have been superseded by a better knowledge.

After having perceived the advantages of the Girod electric heating-system for my laboratory research-work, I was glad to accept several invitations of Director Paul Girod to visit, a few years ago, his old works owned by the Société Anonyme Electrométallurgique Procédés Paul Girod, and, to-day, the new steel-works owned by the Compagnie des Forges et Aciéries Electriques Paul Girod, both at Ugine, Savoie, France, especially since I received permission to make free use of my studies at Ugine for my lectures and publications.

The Girod furnace is the simplest of modern electric steel-furnaces. It is useful both as an experimental furnace and as a smelting-works furnace on a large scale. I am using it in my laboratory with currents of from 30 to 36 kw., and for steel-smelting it is being used with currents of 300 and 1,200 kw. It is a combination of resistance and arc heating-furnace. The metal to be smelted, which acts as one of the electrodes, is covered by the refining-slag as an electrolytic conductor, and a carbon block (or several carbon blocks), entering from above through the cover of the furnace, forms the opposite electrode above the center portion of the bath. The arcs play between the carbon blocks and the slag, and no doubt the largest amount of heat is here produced. A second heat-producing place is the

[1] Borchers, *Electro-Metallurgie*, 3d ed., p. 536 (1903).

layer of slag, through which the current passes to and from the metal. Last but not least, the metal itself becomes an important heat-producer on account of the effective manner in which the electric current is made to pass through the molten metallic body, and this is the most important feature of the Girod process.

The distribution of the electric current through the molten metal is clearly shown in Figs. 1 and 2, in which *A* is the

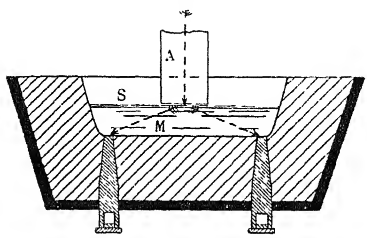


FIG. 1.—SECTION OF GIROD FURNACE, SHOWING THE DISTRIBUTION OF THE ELECTRIC CURRENT.

carbon electrode; *S*, the slag; *M*, the molten metal; and *C*, contact-pieces to connect the metal with the current-conductor. The contact-pieces are of such cross-section and length that each one will take up only a certain part of the whole current without being excessively heated, and therefore without exceed-

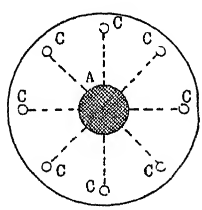


FIG. 2.—PLAN OF GIROD FURNACE, SHOWING THE DISTRIBUTION OF THE ELECTRIC CURRENT.

ingly increasing its resistance. An additional regulation of the temperature and resistance of each contact-piece is effected by cooling-water circulating through small chambers in the end. The contact-pieces are made of pure iron in order to avoid any deterioration of the furnace-charge. They act as the connecting-rods between the furnace-charge and the current-generators, and also regulate the distribution of the current, so that the electric charges pass uniformly from and to the centrally-

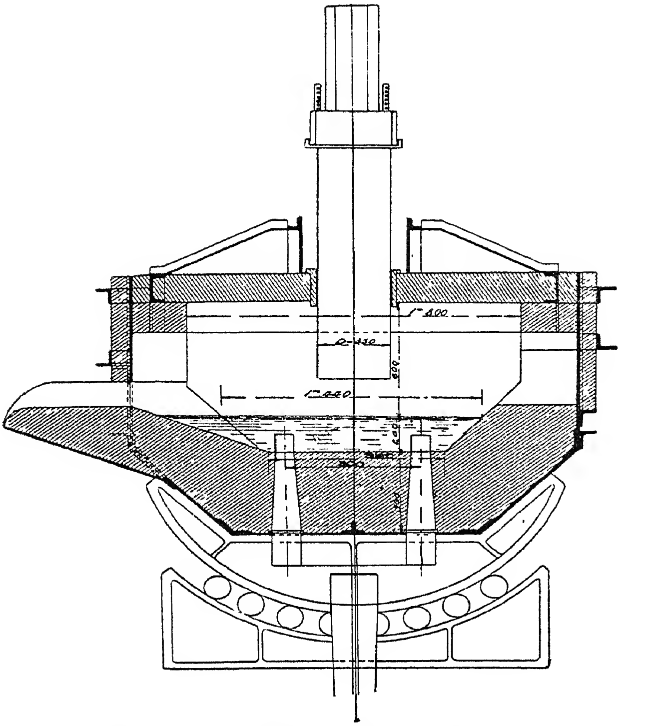


FIG. 3.—SECTION OF 2.5-TON GIROD ELECTRIC FURNACE.

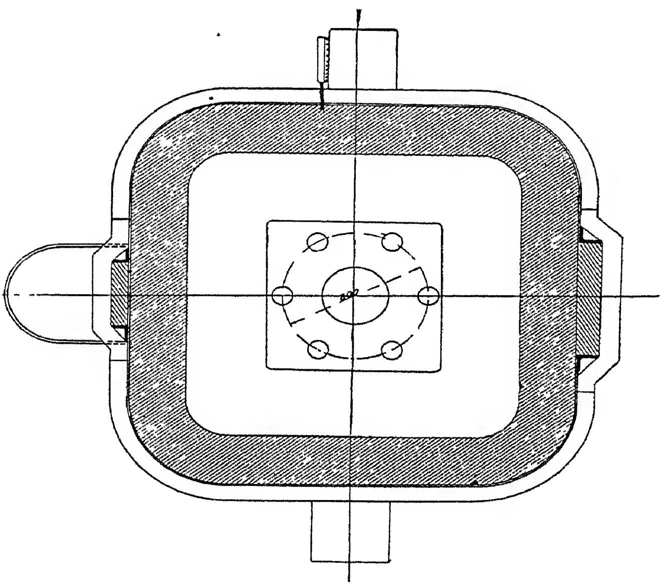


FIG. 4.—PLAN OF 2.5-TON GIROD ELECTRIC FURNACE.

hanging carbon rod or rods in radial direction to and from the periphery of the bath. This condition is important, not only for uniformly heating the bath, but also for keeping every part of the liquid metal in constant motion. Running a furnace with as high a current-density as is found necessary to smelt iron and steel, one can easily observe that the passage and the transformation of the electric current into heat are accompanied by a violent mechanical motion of all particles of the liquid. This movement, however, accelerates the contact between the

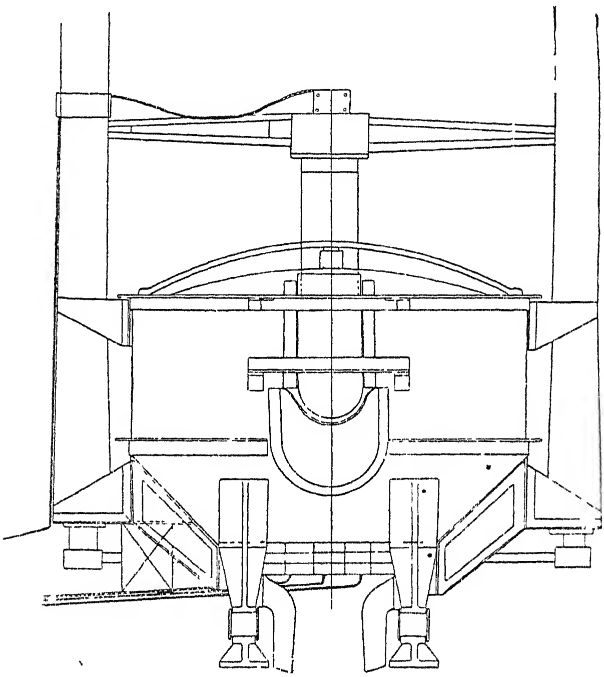


FIG. 5.—SECTIONAL ELEVATION OF 2.5-TON GIROD ELECTRIC FURNACE.

impurities of the iron and the refining-slag which floats on top of the bath. The Girod system of arranging the electrodes and contacts is a safeguard against any stagnation of parts of the liquid furnace-charge, and for this reason the advantages of both the induction or transformer furnaces and the electrode furnaces are combined in the Girod furnace, while the imperfections of either type have been overcome.

The arrangement of electrodes and contacts in the Girod furnace gives it a special advantage over other electrode fur-

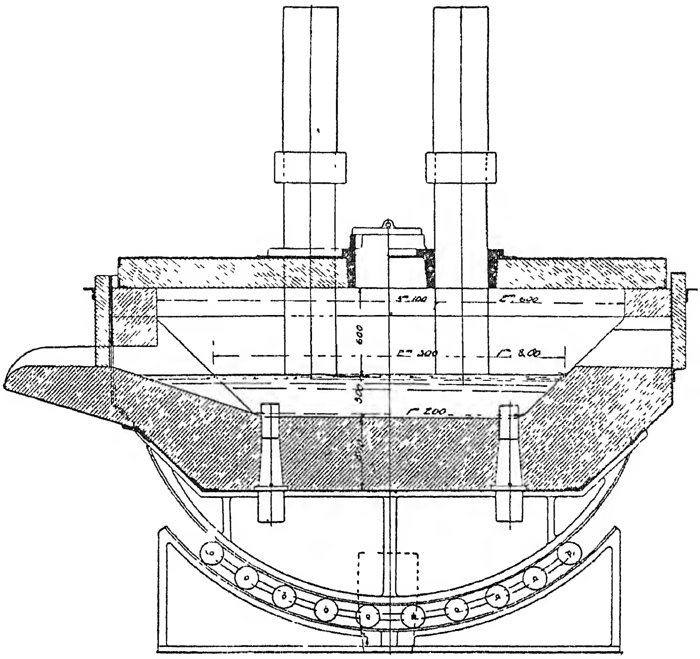


FIG. 6.—SECTION OF 12-TON GIROD ELECTRIC FURNACE.

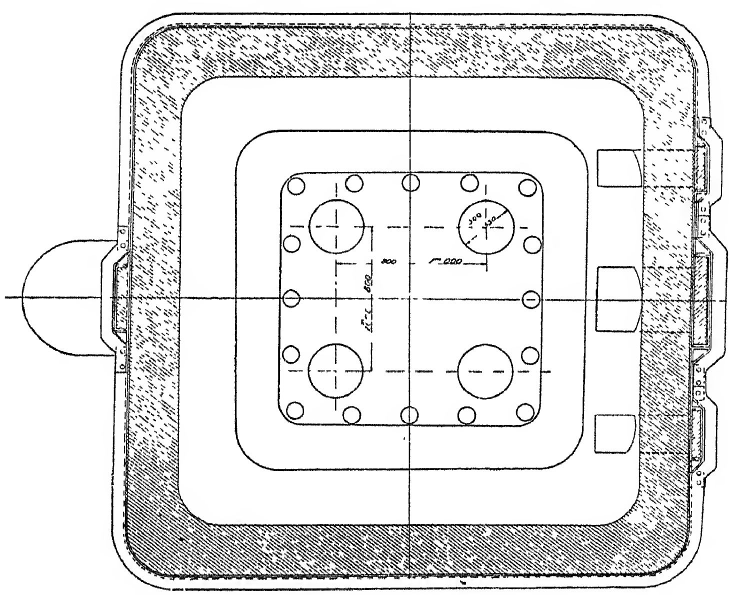


FIG. 7.—PLAN OF 12-TON GIROD ELECTRIC FURNACE.

naces during the melting of cold scrap. As soon as the charge is on the hearth of the furnace, the upper electrode is lowered until it rests upon the heap of scrap; the path of the current is from the center of the heap, by means of numerous small arcs through the whole mass of scrap, to the periphery of the heap on the hearth-bottom. The heap is breaking down simultaneously in all parts in a very short time. There is no sticking of cold pieces to the bottom of the furnace, and therefore no necessity to stir up and loosen such lumps from the hearth, which would endanger the latter.

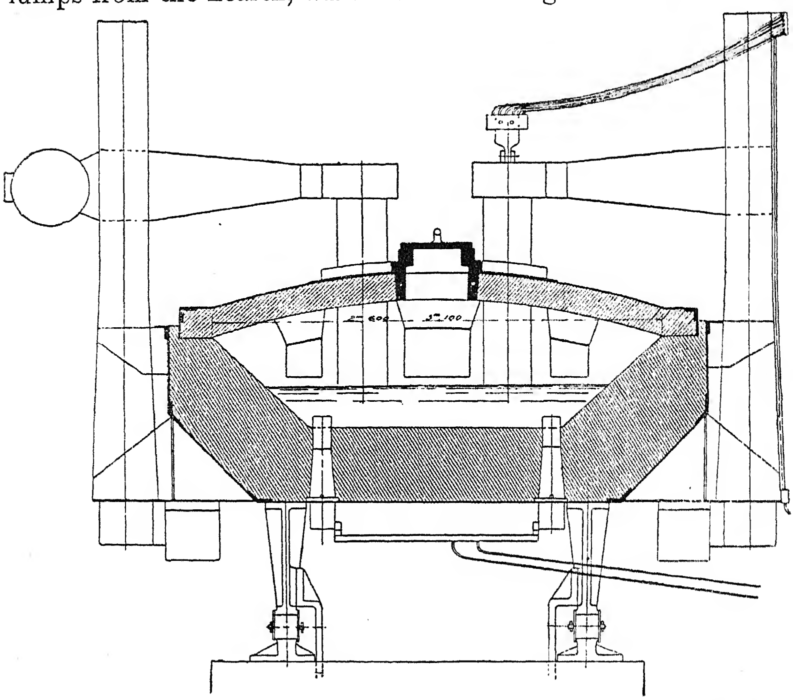


FIG. 8 .—SECTIONAL ELEVATION OF 12-TON GIROD ELECTRIC FURNACE.

The sketches of the Girod furnace, Figs. 3 to 8, illustrate the construction. Figs. 3, 4, and 5 are plans and sections of a 2.5-ton furnace; Figs. 6, 7, and 8, of a 12-ton furnace. The most important dimensions are given in the metric system.

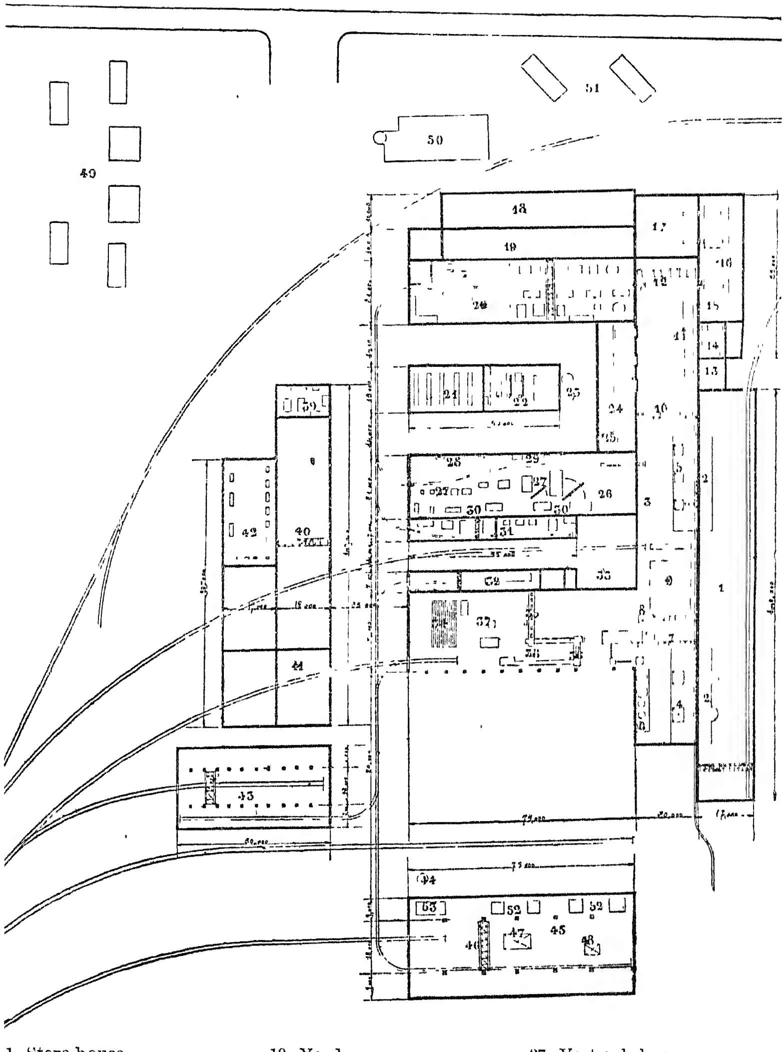
During a smelting-operation, if pieces of the ends of the contact bodies melt and dissolve in the steel, no harm will be done to the latter, since the contacts consist of pure iron, and as the quantity of iron, which in such a case combines with the liquid charge of the furnace, is proportionally so small that the

qualities of the charge will not be influenced thereby to any perceptible degree.

The applicability of the Girod furnaces is not restricted by the nature of the raw materials, which may be either cold scrap or molten metal. In feeding cold scrap the whole charge is not added at once. After the larger part of the charge has been shoveled or otherwise thrown upon the hearth, the current is sent through the heap in the manner already described. The rest of the charge is put into the furnace, together with the first batch of the refining ingredients. Taking the run of a 2-ton furnace as an example, the charge consists of from 2,000 to 2,500 kg. of iron scrap, and the first batch of refining-slag usually consists of about 80 kg. of lime, CaO, and from 220 to 250 kg. of iron oxide ore. Together with the iron oxide that covers the scrap, the iron-ore serves as an oxidizing-agent. The smelting of the iron charge and the first batch of refining-slag requires from 4.5 to 5 hr. This slag becoming exhausted of iron oxide, and therefore of its oxidizing-power, samples are taken and tested to ascertain the degree of refining of the molten metal. According to the degree of purification, the furnace, after the first slag has been skimmed off, receives a second, and if necessary a third, batch of lime-iron oxides. After the removal of the last slag the surface of the metal bath is thoroughly cleansed by adding about 30 or 40 kg. of lime, which is later skimmed off. The further treatment of the iron bath depends upon the absence or presence of impurities which could not be removed by the lime-iron oxide refining and upon the quality of steel to be produced. According to these circumstances, new deoxidizing or otherwise-refining agents are employed; for instance, ferro-mangano-silicon, ferro-aluminum-silicon, ferro-mangano-aluminum-silicon, or other alloys. For producing carbon-steels, an addition is made of Swedish charcoal-iron, or an iron very rich in carbon, produced in an electric furnace at the steel-works.

After the refining-operation, the final step in the production of special steels is the addition of iron-alloys containing nickel, tungsten, chromium, or other special metals.

It is not the purpose of this paper to give a report on researches on the reactions during the refining-process; I shall have to leave this to the special sidero-metallurgists. I wish to



1. Store-house.
2. Furnace-platform.
3. Steel-works.
4. Furnaces.
5. Furnaces.
6. Ladle-heating room.
7. 10-ton crane.
8. 4-ton crane.
9. Casting-pit.
10. 30-ton crane.
11. Molding-machines.
12. 4-ton crane.
13. Contact-tower.
14. Molding-room.
15. Cooling-room.
16. Chimney.
17. Shop for cleaning castings.
18. Lathe-work shop.
19. Yard.
20. Machine-shop.
21. Dressing-room.
22. Power-house
23. Accumulator.
24. Drying-furnace room.
25. Compressor.
26. Forge.
27. Hammers.
28. Hammers.
29. Forge
30. Heating-furnaces.
31. Annealing-furnaces.
32. Roll store-room.
33. Chimney.
34. Cooling-tables.
35. Finishing-mill.
36. Rolling-mills.
37. Vertical shear.
38. 800-h-p. motor
39. Testing-room.
40. 3-ton crane.
41. Shipping-room.
42. Pattern-shop.
43. Tempering-room.
44. Chimney.
45. Forge.
46. 60-ton crane.
47. 10-ton hammer.
48. Forging-press.
49. Laborers' village.
50. Offices.
51. Dwelling-house.
52. Annealing-furnaces.
53. 600-h-p. air-compressor.

FIG. 13.—STEEL-WORKS OF THE COMPAGNIE DES FORGES ET ACIÉRIES PAUL GIROD AT UGINE.

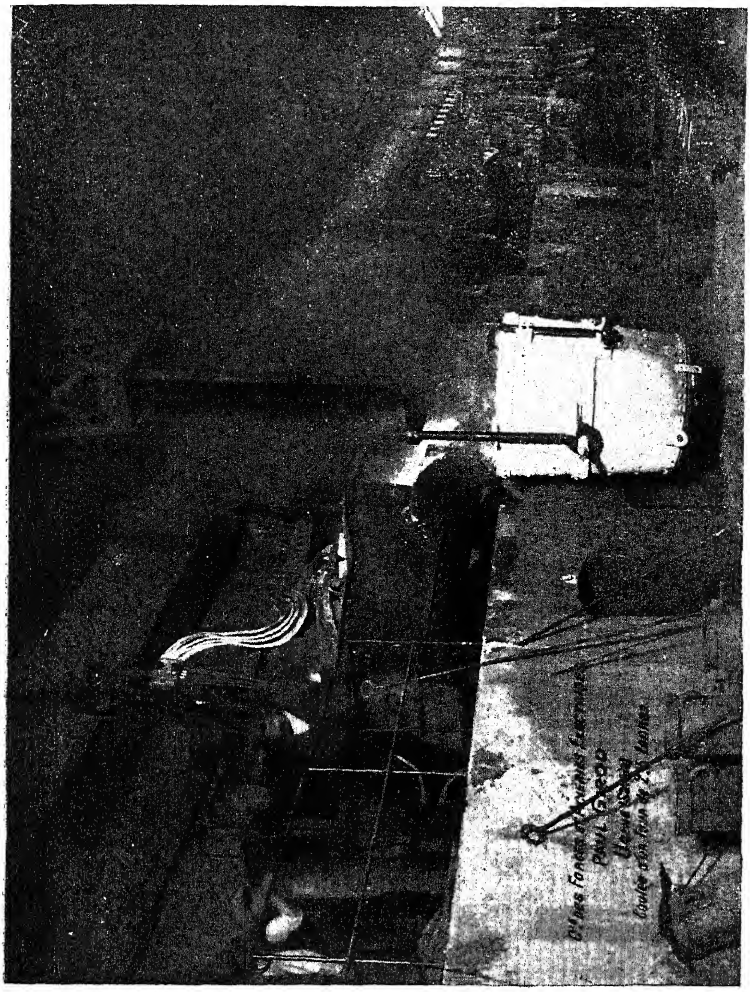


FIG. 9.—2.5-TON FURNACE AT UGINE.

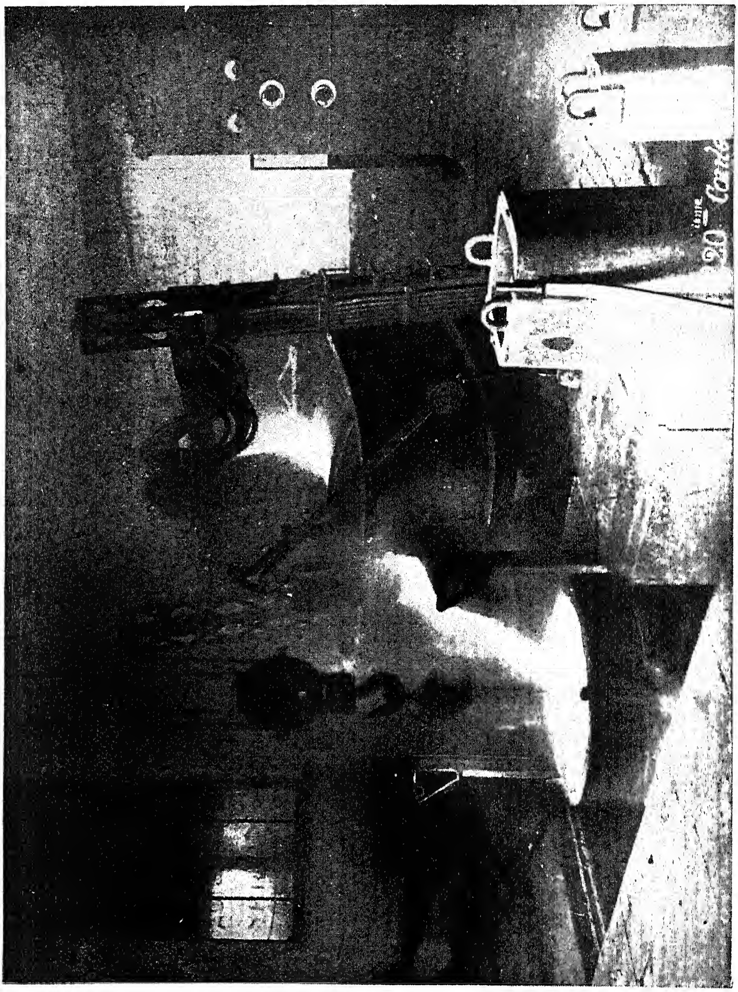


FIG. 10.—2.5-TON FURNACE AT SERAING, BELGIUM, SOCIÉTÉ JOHN COCKERILL.

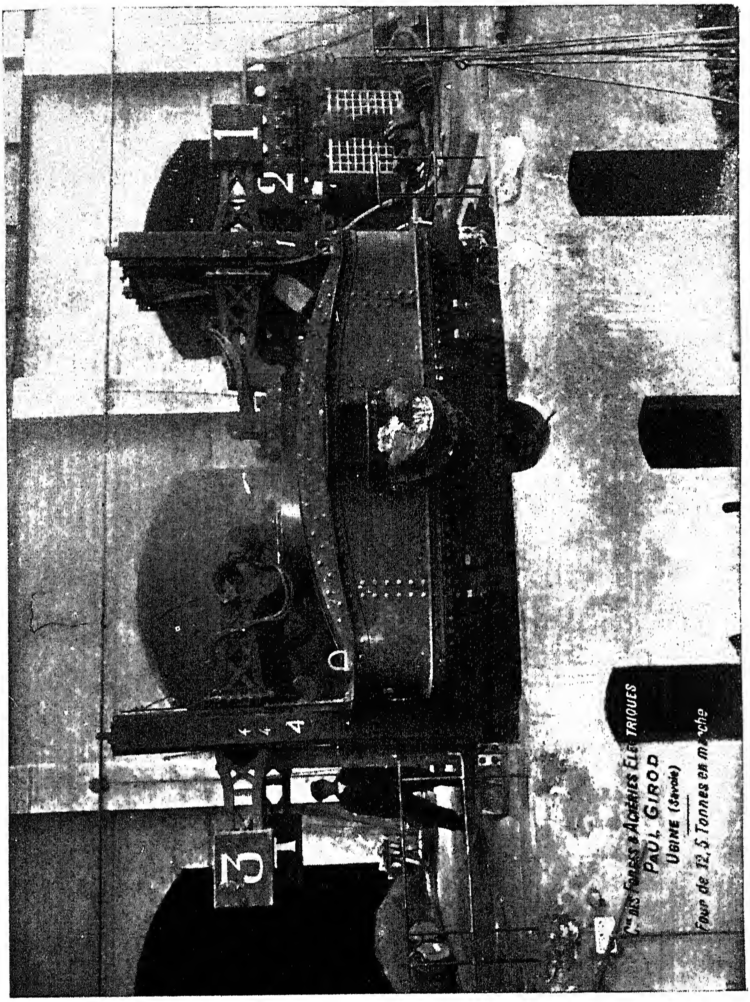


Fig. 11.—12.5-Ton Furnace at Ugine During the Smelting-Operation.

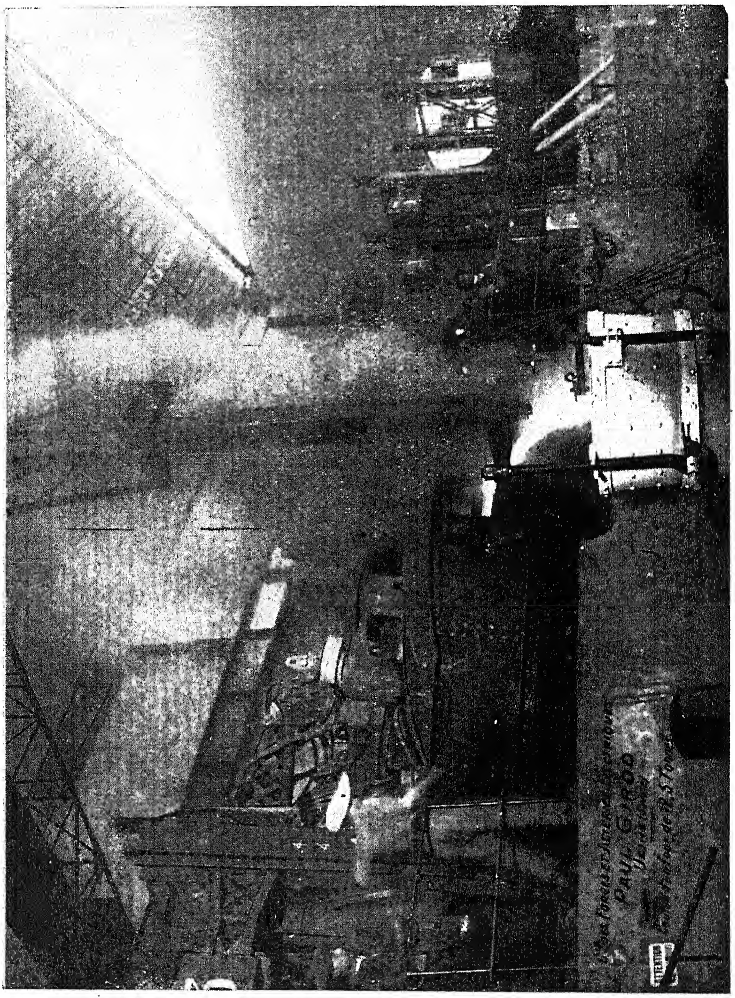


Fig. 12.—12.5-Ton Furnace at Ugine. Finish.

deal to-day with only the practical working of this useful electro-metallurgical apparatus.

The total duration of one smelting-operation, from the starting of the current to the casting of the products into ingots or molds, with raw material that is not pure, covers about 8 hr. With pure raw material less time and power will be required.

The smaller of the two types of furnaces used in steel-works is run with alternating currents of about 300 kw. at from 60 to 65 volts. The larger requires from 1,000 to 1,200 kw. at from 70 to 75 volts. Taking into account a loss of 10 or 11 per cent. of the charge by oxidation and evaporation, the power required to make 1 ton of steel in a small furnace is from 900 to 1,000 kw-hr., and from 800 to 900 kw-hr. in a large one.

The consumption of carbon electrodes amounts to from 12 to 15 kg. per ton of steel, including the short pieces which remain in the holders when the latter have reached the lowest point at which they can withstand the radiating heat of the bath.

The lining of the furnace, consisting mostly of calcined dolomite, will withstand at least 80 charges; but cavities eaten into the side-walls by the slag around the level of the molten bath are repaired with calcined dolomite at shorter intervals of time. The bottom of the furnace will stand from 120 to 160 charges without repair. The corrosion of the bottom during this time is about 100 mm.

Furnace-linings will stand a much longer time if liquid charges are used. One of these furnaces has run 200 charges without any repair.

The linings of the furnace-covers require more frequent renewing: in the small furnaces, after running 25 or 30 charges, and in the larger furnaces, after 20 or 25 charges.

The other parts of the furnace outfit are not worn out as fast as the linings and electrodes; the furnaces which so far have been put into operation still have their original outfits.

At the works at Ugine, scrap-iron is the only raw material used for steel-making as well as for the manufacture of iron-alloys. The larger part of the scrap is soft iron and steel, the smaller part pig- and cast-iron. The average percentage of impurities in the raw material used during a long period of operations was:

	Per Cent.
C,	0.400 to 0.500
Si,	0.150 to 0.250
Mn,	0.500 to 0.700
S,	0.060 to 0.090
P,	0.080 to 0.100

The degree of refining which can be reached in the Girod furnace is shown in Table I.

TABLE I.—*Analyses of Various Steel-Works Products.*

No.	Properties.	C. Per Cent	Si. Per Cent.	Mn. Per Cent.	S. Per Cent.	P. Per Cent.	Other Constituents. Per Cent.
1.	Very soft	0.079	0.106	0.205	0.015	0.012	......
2.	Soft	0.233	0.180	0.431	0.012	0.010	.....
3.	Middle soft	0.283	0.208	0.430	0.014	0.010	......
4.	Middle hard	0.388	0.155	0.342	0.011	0.009	......
5.	Middle hard	0.463	0.204	0.463	0.010	0.016	......
6.	Hard	0.596	0.198	0.302	0.017	0.005	......
7.	2 per cent. of nickel	0.076	0.099	0.101	0.014	0.010	2.12 Ni
8.	3 per cent. of nickel, soft	0.060	0.123	0.209	0.013	0.007	3.47 Ni
9.	3 per cent. of nickel, hard	0.364	0.144	0.435	0.012	0.015	3.41 Ni
10.	5 per cent. of nickel, soft	0.134	0.148	0.375	0.016	0.013	5.25 Ni
11.	5 per cent. of nickel, middle soft	0.250	0.157	0.414	0.010	0.015	5.08 Ni
12.	Nickel-chrome	0.420	0.199	0.500	0.010	0.009	2.53 Ni 0.77 Cr
13.	Tool-steel	1.223	0.168	0.224	0.011	0.010	......
14.	Tool-steel	1.474	0.119	0.264	0.015	0.007	......
15.	Tool-steel	1.010	0.219	0.306	0.008	0.009	0.32 Cr
16.	Tool-steel	1.277	0.230	0.130	0.009	0.006	0.24 Cr
17.	Tool-steel	1.251	0.176	0.258	0.010	0.008	1.21 Cr 0.49 Ni
18.	Tool-steel	0.689	0.029	0.096	0.012	0.009	6.07 Cr 0.46 Mo 25.82 W

The cost of the different types of Girod furnaces and of complete plants is as follows:

A 2.5-ton furnace, including regulators for the electrodes, measuring-instruments, tilting-device mechanism, and conductors from the furnace to a dynamo, or transformer, near the furnace-room, costs about 15,000 francs.

A 12.5-ton furnace with a similar outfit costs about 30,000 francs.

The manufacturing-costs of the electrodes amount to 3 or 4 francs per ton of carbon body.

A complete plant, with one 2.5-ton furnace for regular running, and one furnace for reserve, with all appliances and smelter-building, but without dynamo or transformer, will cost approximately from 200,000 to 300,000 francs.

A plant equipped with a 12.5-ton furnace will cost from 300,000 to 400,000 francs.

Several years ago the Société Anonyme Electrométallurgique Procédés Paul Girod, at Ugine, Savoie, France, was organized to operate the Girod processes for making iron-alloys and steel. This company owns the Girod alloy-works, in which also the first experimental 2-ton steel-furnace was tried. The fact that to-day 19 furnaces with from 400 to 600 e.h-p. are in operation, and that 12 new furnaces of 1,200 e.h-p. each are being erected, shows the extent of this manufacture. In the future these old works will be limited to making alloys only, and the old steel-furnace will be transferred to the new steel-works.

This company owns also an electrode factory with the most perfect and modern machinery, of a capacity of from 10 to 12 tons of electrode-carbons, round or square in cross-section, of diameters up to 350 mm., and in lengths up to 1,600 mm. At present only 7 tons are made, for which one furnace is used; a second furnace will be built as soon as the demand for electrodes increases.

The raw materials for the electrodes are retort-graphite, petroleum-coke, anthracite coal, and anhydrous tar. The solid materials are ground in ball-mills, kneaded with tar in double-screw mixers and in Chilean mills. The scarcely plastic mixture is now stamped into big blocks, which finally are pressed into blocks of the dimensions given above. These blocks are burned in gas-fired ring-furnaces, in the chambers of which are large fire-clay tubes containing the pressed blocks under a covering of ground coal.

The company owns several water-powers. The plant built for the old works is situated about 3 km. above Ugine, on the Arly river. Here a strong dam with gates sends the water into a tunnel, which leads it to a pipe-line directly above the power-house at Ugine. Ten Pelton-wheel turbines drive as

many directly-coupled dynamos, which furnish a total of from 8,000 to 9,000 e.h-p. The second power-station, furnishing an additional 12,000 e.h-p., is situated above Le Fayet. The tunnel and pipe-lines are large enough to carry the water for 20,000 e.h-p., but the power-house is equipped with turbines and dynamos for only 12,000 e.h-p. It will soon be enlarged. The three-phase current is sent under 45,000 volts to Ugine, where it arrives with from 40,000 to 42,000 volts. It is transformed here into currents of 2,500 volts for the large machines, and of from 60 to 75 volts for the furnaces. The 2,500-volt current is transformed into a current of 500 volts, and this into 110-volts for light and small motors, and a 500-volt direct current for locomotives, cranes, and similar uses.

In addition to this power the company leases from a power-station at Bionnay, upon the Bonnant river, near St. Gervais, 6,000 e.h-p., and from the Société d'Electrométallurgie Sud-Est at Veuthon, near Albertville, during the period October 1 to April 1, from 8,000 to 10,000 e.h-p. From both stations the current is furnished under an e.m.f. of 45,000 volts. The current from St. Gervais is conducted to Le Fayet, where it enters the Le Fayet-Ugine line.

For the production and the first mechanical and thermal treatment of the special steels, a new company, the Compagnie des Forges et Aciéries Electriques Paul Girod, also under the management of Paul Girod, has been formed. The works of this company have been built about 1 km. below the old works, and the electric current is supplied by the old company.

The installation which has been provided for the new steel-works, most of which is now in running order, is the following: Three 2-ton electric furnaces, one of which is used for small steel castings, and two 10- to 12-ton electric furnaces. The steel produced in these furnaces, except the one mentioned, is cast by means of large ladles, worked by electric cranes, into ingots, which go to the rolling-mills. Working with several furnaces for steel castings, pieces (dynamo-frames) up to 20,000 kg. have been made.

For the mechanical and thermal treatment of the steel ingots, the best modern machinery has been erected.

The rolling-mill has 2 three-high rolls, one of which can roll ingots of 525 mm. diameter and 400 kg. weight to rods of

125 mm. diameter. The other rolls ingots of 325 mm. diameter to smaller round, square, and other shaped rods. This mill has two furnaces for heating the ingots before passing to rolls, and the necessary cutting- and sawing-machinery for the rolled shapes.

A large forging-shop contains 9 hammers operated by compressed air. The largest of these hammers weighs 5,000 kg.: the weights of the other 8 hammers vary from 1,000 to 100 kg. Here specially rough forgings are made, primarily for automobile machinery, shaftings, gears, tool-steel rods, projectiles, etc.

In addition to several annealing-furnaces already in operation, a large shop is being built for tempering, annealing, and hardening large forged or cast-steel pieces.

A second forging-shop, now being erected, is to be equipped with one 1,000-ton forging-press, one forging-hammer of 10 tons hammer-weight, and three stamps with falling hammers, weighing respectively 3,000 kg., 2,000 kg., and 1,000 kg.

The steel-foundry contains a carpenter-shop for making patterns, a sand-separating plant, molding-machines, molding-frames, drying- and heating-furnaces, and a sand-blast jet for cleaning the finished castings. The foundry, with its present equipment, will be able to produce about 10 tons of steel castings daily, but, as mentioned above, pieces weighing up to 20,000 kg. may be cast.

A machine-shop now nearly completed has 6 drills, 5 lathes, 2 planers, and 1 grinder. Each machine is operated by a special motor.

The power-house is equipped with two transformers, two commutators, and four air-compressors, one of 30 h-p., one of 40 h-p., and two of 400 h-p. capacity.

The analytical laboratory, situated in the old works, is doing the investigations for both companies, and employs 15 chemists and assistants.

The new plant has a laboratory for making mechanical and physical tests of the products, which is equipped with all appliances to determine the mechanical strength and to investigate the microstructure as changed by mechanical and heat-treatment.

The two companies are erecting sanitary and comfortable dwelling-houses for the workmen and other employees.

I have been informed that in addition to the two Girod companies mentioned above, the following firms have ordered Girod furnaces, some of which have been operated for some time:

Oehler & Cie., steel-foundry in Aarau, Switzerland.
Société John Cockerill, iron- and steel-works in Seraing, Belgium.
Stotz & Co., steel-foundry in Kornwestheim, near Stuttgart, Germany.
Stahlwerk Becker, Krefeld, Germany.
Gutehoffnungshutte (one of the largest and oldest coal-mining, iron- and steel-smelting works in Germany), in Oberhausen.
Ternitzer Stahlwerk, Schoeller, in Ternitz, Austria.
Danner & Co., steel-works in Judenburg, Austria.
Ungarisches Staats-Stahlwerk in Diosgyor, Hungary.
Fried. Krupp, A.-G., Steel Foundry in Essen-Ruhr, Germany.

From the above it is evident that at last the Girod furnace is earning the attention and success it so well deserves.

The Ultimate Source of Ores.

BY CHARLES R. KEYES, DES MOINES, IOWA.

(Pittsburg Meeting, March, 1910.)

CONTENTS.

I. INTRODUCTION.

If the long controversy waged between those who, on the one hand, advocated a strictly deep-seated origin of ore-deposits, and, on the other hand, those who argued for an ore-genesis by

the leaching of near-by rocks, had had no other result than to bring out from obscurity three certain features of practical import, all the labor of that controversy would have been well expended.

These three points are: (1) a clear distinction between the vadose and the profound zones; (2) a determination of the manner in which the latter may be invaded by conditions of the former; and (3) an establishment of the general truth of the lateral-secretion theory when vadose conditions prevail, and of the ascension-theory under profound conditions. In the practical consideration of ore-deposits these generalizations are fundamental. Their wider application must be productive of great industrial advantages.

Of first importance in the broader scientific survey of ore-genesis is the recognition of a sharp distinction between vadose and profound conditions. To many who listened to Professor Posepny's remarkable paper, read at the Chicago session of the Institute during the Columbian Exposition in 1893, this seemed, as Dr. S. F. Emmons has already observed, its most valuable and suggestive feature. Notwithstanding Posepny's implications to the contrary, and his vigorous berating of Sandberger's views, the immediate and lasting effect of the argument was really to strengthen greatly Sandberger's position. For it completely and sharply separated the lateral-secretionists from the ascensionists, raising the sphere of the former bodily above the line of permanent water-level and at the same time confining the sphere of the latter to regions below that horizon.

A little later, Professor Van Hise showed in some detail how meteoric waters may deeply invade the profound zone and carry vadose conditions into it.

The zonal limitations of two great and apparently opposed agencies being thus definitely fixed, it is found that neither of the theories based thereon is susceptible of general application. If the generalizations of the lateral-secretionist be propounded solely for vadose conditions, we are able to agree with all of his more-important contentions; while there is general acquiescence in the views of the ascensionist if their application be restricted to the profound zone.

Of the ore-bodies of the world that are mined to-day, the majority are essentially vadose in origin, and probably originated

mainly according to the principles advanced by the advocates of the lateral-secretion theory. What proportion of worked ore-deposits owe their segregation directly and entirely to geologically deep-seated influences is, in the light of recent investigations, difficult to estimate.

Since, chemically, all ores may be regarded as deposited in one way or other from solution (apart from the special cases of magmatic secretions from molten rock-solution, and of placers, as the heavy residues after gangue and country-rock have been dissolved and carried away), the consideration of ore-origin is greatly simplified.

II. Four Phases of Primary Ore-Genesis.

However opinions may differ concerning the immediate derivation of lode-materials, there seems to be at present a general tendency to regard the igneous rocks as, directly or indirectly, the ultimate source of the metals. In this consideration the view-point is not always the same. When carefully analyzed a half-score of decided variations are found. In some cases several are combined. In others special stress is laid upon some particular feature.

These and other various phases of primary ore-genesis may be all reduced to four principal groups: (1) extraction from sea-water; (2) inclusion of metallic minerals as accessories in the igneous rocks themselves and the subsequent extraction and segregation of the ore-materials through weathering-processes; (3) production of metalliferous bodies in connection with rock-masses in a molten state, either through magmatic secretion or by the expulsion of the volatile compounds of the metals during the process of magma-cooling; and (4) derivation of metallic particles from extra-terrestrial sources, and their later segregation through the action of surface-water.

Of these several aspects of ore-genesis, the first is now obsolete; the second and third enter into nearly all of the recent discussions of the subject; the last mentioned receives yet only incidental attention.

1. *Extraction of Ore-Materials from Sea-Water.*

A. Early Views.—The old notion that the origin of ore-materials is in the water of the ocean must be regarded as a theo-

retical surmise rather than as a scientific deduction based upon observed facts. The eminent chemist Bischof[1] especially, among other distinguished writers, early argued for sea-water as the primitive source of the metallic salts in nature.

The belief that metallic salts from the ocean were largely gathered into ore-bodies in marine sediments when the latter were laid down, had its foundation in the erroneous assumption that rock-masses once formed subsequently undergo no change. From this premise it was easy to draw the conclusion that ores in Silurian rocks, for instance, were deposited during the Silurian period. Moreover, even granites, gneisses, and schists were generally regarded as only highly-metamorphosed sediments.

B. Metals in Sea-Water.—Since the discovery of the existence of the precious metals in sea-water by Malaguti, Durocher, and Sonstadt,[2] and especially the later detection by Forchhammer[3] of all of the more common metals, many facts have been cited in support of this being the original source of ore-materials. In the same way have been quoted Dittmar's elaborate investigations[4] on the chemistry of sea-water. Altogether, 32 chemical elements, including all the common metals, have been detected in appreciable quantities in sea-water; and it is believed that all the known chemical elements occur in it.

C. Metallic Content of Sedimentary Rocks.—Some recent utterances bear more directly upon the primal derivation of ore-materials. Van Hise and Bain, in particular, have argued it for the lead- and zinc-deposits of the Mississippi valley,[5] and the former has given it general application.[6] These authors call the general dissemination to the extent of a minute fraction of 1 per cent. of the metal through the sedimentary rocks a "first concentration."

It would be difficult to secure decisive proof either for or against the original inclusion of appreciable quantities of metallic minerals during marine sedimentation. At most, such

[1] *Chemische und physikalische Geologie,* vol. i., p. 836 (1847).

[2] *Chemical News,* vol. xxvi., p. 159 (1872).

[3] *Philosphical Transactions of the Royal Society of London,* vol. clv., p. 203 (1865).

[4] *Reports on the Scientific Results of the Voyage of H. M. S. Challenger,* vol. i., Physics and Chemistry (1884).

[5] *Transactions of the Institution of Mining Engineers,* vol. xxiii., p. 576 (1901–02).

[6] *Monograph No. XLVII., U. S. Geological Survey,* p. 1030 (1905).

a process would be one of general diffusion rather than of incipient concentration. Even granting the premises of a comprehensive circulation of ground-water, it might be seriously questioned whether any metallic content of such rocks was not introduced long after the sediments themselves were laid down. It may also be asked whether the metallic content could not be several times, or even many times, extracted and returned through the same means.

Special chemical analyses of sedimentary rocks in mining-districts indicate clearly that the amounts of the metals contained are amply sufficient to supply the materials for the local ore-bodies many times over. But too great confidence should not be placed upon these data as proofs of an original deposition of the ore-materials from the sea. Analyses of sedimentary or metamorphic rocks have not the same elements of certainty as those of igneous rocks. There is no known test whereby the exact original character of the clastics can be shown. The results of large-quantity analyses of the Missouri limestones and dolomites by Robertson[7] are as accurate as are those of the analyses of the igneous rocks by the same chemist;[8] yet in the latter case the microscope clearly shows the unaltered character of the samples taken, while in the former case there is every reason to believe that the samples of limestones, coming from the vadose zone, have been more or less extensively altered. The same is true of Weems's analyses[9] of the Ordovician rocks of the Dubuque lead-district in Iowa. Being notably porous, conspicuously jointed, severely fractured, frequently seamed and often pre-eminently cavernous, besides being in the vadose zone, these rocks have given every facility for the ready introduction of mineral-bearing waters, and the precipitation of minute grains of metallic minerals throughout the entire mass.

D. Adequacy of Metal-Percentage to Form Ore-Bodies.—There seems to be little doubt that sea-water, marine sediments, and metamorphic rocks contain ample amounts of the metals to supply the most extensive local ore-deposits. For instance, Liversidge's estimate,[10] allowing only from 2 to 5 mg. of gold to the metric ton of sea-water, gives for the ocean the enormous

[7] *Missouri Geological Survey*, vol. vii., p. 483 (1894). [8] *Ibid.*, p. 478.

[9] *Iowa Geological Survey*, vol. x., p. 567 (1900).

[10] *Journal of the Royal Society of New South Wales*, vol. xxix., p. 335 (1895).

gold-content of one hundred and fifty billion metric tons. Buell[11] calculates that in the Potosi district, the most productive lead-area of Wisconsin, there is 1–1400 of 1 per cent. of metal in the rocks, limiting the volumes to a layer 100 ft. in thickness and taking into account only one-half of the distance between the crevices or veins. This is scarcely one seven-millionth part of the rock-mass.

In the limestones of the Ozark region of Missouri, Robertson[12] found the average content of lead to be more than 0.001 per cent., and of zinc more than twice as much. A block of limestone one mile square and 500 ft. thick would thus yield about 14,000 tons of metallic lead and 42,000 tons of metallic zinc. In the testing of hundreds of rocks of all kinds, Dieulafait[13] was always able to detect readily appreciable quantities of copper and zinc. A large series of metamorphic rocks from British Guiana yielded to Harrison[14] notable quantities of copper, and often of other heavy metals. In all these cases, the diffused metals are claimed to have been incorporated at the time the sediments were laid down.

E. Derivation of Metallic Elements in Sea-Water.—The metallic content of the ocean must be regarded as chiefly, if not only, secondary in character. A large part is manifestly derived directly from the land-waste brought in by the rivers; a smaller portion is dissolved out of the volcanic dust falling into the water; and a third, and perhaps the greatest, part is derived from star-dust raining down upon the water-areas of the globe.

So far as ore-deposition is concerned, the occurrence of ore-materials dissolved in the water of the sea must be viewed as a process of diffusion rather than as one of even incipient concentration. The same is true of the solvent effect of sea-water occluded in marine sediments. Although there are always ample quantities of ore-materials in both the sea and its sediments, the influence of circulating ground-water in the rocks after their deposition probably preponderates so greatly over other ore-forming agencies that, as a primary source of the

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11 *Geology of Wisconsin*, vol. iv., p. 538 (1882).

12 *Missouri Geological Survey*, vol. vii., p. 480 (1894).

13 *Annales de Chimie et de Physique*, Fifth Series, vol. xviii., p. 349 (1879).

14 *Report on Petrography of Cuyani and Mazaruni Districts, Georgetown, Demerara* (1905).

ores, the amounts originally deposited on the old sea-floors may be entirely neglected. The metal-content of the sea being a phenomenon of diffusion rather than of concentration, it does not affect, to any appreciable extent, the localization of important ore-bodies. The peculiarities of geological structure in all mining-regions seem amply sufficient to account for local ore-segregations.

2. *Metallic Content of Igneous Rocks.*

A. Two-fold Meaning of Term.—Regarding the derivation of lode-materials from the igneous rocks, clear distinction is not always made between the two physical states in which rock-masses occur. Ore-segregation from already solidified igneous masses involves processes and conditions very different from those of a cooling rock-magma where the metallic compounds are being expelled in volatile form and are collecting around the margins of the molten mass. In some discussions of ore-genesis one of these conditions is alone considered; in others, only the other.

Igneous masses already cooled and solidified were specifically referred to by Sandberger.[15] Van Hise[16] manifestly uses the terms "igneous mass" and "magma" as synonyms, when he advocates the propositions that igneous rocks are the direct source of some ores; that they are the ultimate source of all ores; and that the heat of igneous rocks is of fundamental importance in the segregation of the ores; and concludes, therefore, that the original source of all ore-deposits is believed to be magma. In the present discussion, "igneous rock" connotes the solidified state only and "magma" the molten condition.

B. Significance of Ordinary Rock-Analyses.—As commonly made, chemical analyses of rocks do not indicate the presence of any of the elements which occur in minute quantity. Hence most of the metals, unless especially sought for, escape detection. The determination of from five to ten simple or compound ingredients is the common practice of chemical laboratories. Of the thousand-odd rock-analyses collected and tabu-

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[15] *Untersuchungen über Erzgänge,* Part II. (Wiesbaden, 1885).

[16] *Monograph No. XLVII., U. S. Geological Survey,* p. 1030 (1904).

lated by Roth,[17] few give any signs of the presence of the heavy metals. Even in the later and more refined analyses of igneous rocks, undertaken mainly for petrographic purposes, in each of which a score of determinations are made, for such institutions as the United States Geological Survey, the State laboratories, and the laboratories of universities, there is still a remarkable apparent absence of the heavy metals. Yet when these metals are specifically sought, they appear in determinable quantities, as is amply attested by the work of Robertson, Weems, Don, Hillebrand, and others.

C. Wide-Spread Occurrence of Rarer Rock-Forming Minerals.—In the well-known case of the Maryland granites,[18] numerous chemical analyses indicate only the common rock-constituents. Yet several interesting minerals (such as epidote and allanite, the latter of which contains the rare earths cerium, lanthanum, didymium, and yttrium), which the analyses do not suggest, form important accessory compounds. Both epidote and allanite are readily distinguishable by the naked eye, and are easily separated in notable quantities by means of Thoulet's solution. Allanite is also now known to be widely distributed in eruptive rocks, as was shown long ago by Scheerer,[19] and recently by Iddings and Cross,[20] Hobbs,[21] Lacroix,[22] and others.

D. Occurrence of Metalliferous Minerals in Rocks.—The wide-spread occurrence of metallic minerals in the massive crystalline or igneous rocks is now not only known in a general way from chemical analysis, but is readily shown in thin slices of such rocks under the microscope. In view of the early chemical work on rock-composition half a century ago, and the later more elaborate investigations, many petrologists are inclined to ascribe the original source of the metals directly to igneous rocks. Though this view needs qualification, the metallic content of igneous rocks may be still considered as, in some re-

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[17] *Die Gesteins-analysen in tabellarischer Übersicht und mit kritischen Erläuterungen* (1861).

[18] *Fifteenth Annual Report, U. S. Geological Survey*, p. 704 (1893 4).

[19] *Poggendorff's Annalen der Physik und Chemie*, vol. lvi., p. 479 (1842).

[20] *American Journal of Science*, Third Series, vol. xxx., No. 176, p. 108 (Aug., 1885).

[21] *Johns Hopkins University Circulars*, No. 65, p. 70 (1888).

[22] *Bulletin de la Société minéralogique de France*, vol. xii., p. 139 (1889).

spects, directly related to ore-genesis. Professor Van Hise[23] has recently said: "I have little doubt that the metallic constituents of ores are in large part derived from the igneous rocks which have been intruded into or extruded upon the lithosphere."

Specially significant in this connection is the extensive series of chemical analyses of rocks conducted by Forchhammer,[24] in which appreciable amounts of all of the common metals were found. His inference that the metallic content of ore-bodies was derived mainly from the adjoining rocks thus preceded the publication of Sandberger's famous hypothesis by more than a quarter of a century. By still more refined investigations Sandberger[25] traced the principal metallic content of the igneous rocks to their ferro-magnesian constituents. Although this author's views on the leaching of the metals from the iron-bearing minerals have been so vigorously combated, notably by Stelzner[26] and Posepny,[27] the general occurrence of minute quantities of the metals in the igneous rocks is conclusively demonstrated by the work of many careful investigators.

E. Metallic Content of Igneous Rocks Generally.—The earlier results of rock-analyses by Dieulafait,[28] Curtis,[29] Hillebrand,[30] Mallet,[31] and Becker[32] are interesting; but far greater in scientific import and practical value are the results of Robertson,[33] who made conclusive determinations of the lead-, zinc-, and copper-content of Missouri igneous and sedimentary rocks; of Emmons,[34] who showed by L. G. Eakins's analyses the general presence of silver in the igneous rocks of central Colorado; of Don,[35] who made an exhaustive inquiry into the gold-bearing rocks of Australia; of Levat,[36] who determined the gold- and silver-content of the dia-

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23 *Monograph No. XLVII., U. S. Geological Survey*, p. 1030 (1904).

24 *Poggendorff's Annalen der Physik und Chemie*, vol. xcv., p. 60 (1855).

25 *Berg und Hüttenmännische Zeitung*, vol. xxxvi., No. 44, p. 377 (Nov. 2, 1877).

26 *Zeitschrift der deutschen geologischen Gesellschaft*, vol. xxxi., p. 644 (1879).

27 *Trans.*, xxiii., p. 247 (1893).

28 *Annales de Chimie et de Physique*, Fifth Series, vol. xviii., p. 349 (1879).

29 *Monograph No. VII., U. S. Geological Survey*, p. 80 (1884).

30 *Ibid.*, vol. xii., p. 591 (1886).

31 *Chemical News*, vol. lv., No. 1416, p. 17 (Jan. 14, 1887).

32 *Monograph No. XIII., U. S. Geological Survey*, p. 350 (1888).

33 *Missouri Geological Survey*, vol. vii., p. 479 (1894).

34 *Nineteenth Annual Report, U. S. Geological Survey*, Part II., p. 471 (1896).

35 *Trans.*, xxvii., 564 (1897).

36 *Annales des Mines*, Ninth Series, vol. xiii., p. 386 (1898).

bases of French Guiana; of Wagoner,[37] who found relatively large amounts of both gold and silver in the granites, syenites, diabases and other rocks of California and Nevada; of Van Hise,[38] who states that to him the copper-content sparsely distributed throughout the basic Keweenawan lavas of the entire Lake Superior basin has much greater significance with reference to the source of the metal than has the occurrence of the ore with an igneous deposit at a particular point; of Spurr,[39] who determined that in the Monte Cristo mining-district of Washington, the ores are derived "directly by concentration from the tonalite in which they lie;" and of Keyes, who found that in the Gold mountains of New Mexico, comprising the lofty Los Cerrillos, Ortiz, Tuertos, and San Ysidro laccoliths, the fresh mica-andesites often assayed over one dollar a ton in gold. The analyses of two-score or more of igneous and metamorphic rocks of British Guiana gave Harrison [40] relatively high percentages of most of the common metals, while in one sample only was gold absent.

F. Metals in Volcanic Rocks.—Chemical analyses of recent lavas and other volcanic extrusions also show that they have a notable metallic content. Thus in the volcanic ash of Vesuvius nearly 1 per cent. of copper oxide was found by Commanducci.[41] Andesitic lavas of Lautoka, in the Fiji islands, yielded Jensen [42] more than 0.03 per cent. of copper. Mallet [43] found notable amounts of silver in the ash of the volcano Cotopaxi, and of Tunguragua.[44] Numerous other similar results might be cited.

G. Originality of Metal-Content of Rocks.—The determination of the original nature of the metallic content of igneous rocks is especially trustworthy, because, in thin slices under the microscope, it can be told at a glance whether or not the rock has ever suffered any change, whereby the metals were intro-

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[37] *Trans.*, xxx., 798 (1900).

[38] *Monograph No. XLVII.*, *U. S. Geological Survey*, p. 1033 (1904).

[39] *Twenty-second Annual Report, U. S. Geological Survey*, Part II., p. 829 (1900–01).

[40] *Report on Petrography of Cuyuni and Mazuruni Districts, Georgetown, Demerara* (1905).

[41] *Gazzetta chimica italiana*, vol. xxxvi., Part II., p. 797 (1906).

[42] *Chemical News*, vol. xcvi., No. 2503, p. 245 (Nov. 15, 1907).

[43] *Ibid.*, vol. lxv., No. 1416, p. 17 (Jan. 14, 1887).

[44] *Proceedings of the Royal Society of London*, vol. xlvii., p. 277 (1890).

duced subsequently to the solidification of the rock-mass. In this respect, the analyses of metamorphic and sedimentary rocks cannot be always depended upon. Thus, while in the case of the Missouri granites and diabases analyzed by Robertson[45] the results are to be implicitly relied upon, his equally accurate determinations for the limestones and dolomites afford, as already stated, no criterion. Weems's[46] analyses for the lead- and zinc-content of Iowa rocks are similarly inconclusive. These analyses really tell no more than the observations with the naked eye. Moreover, the probabilities are strongly in favor of the view that the metallic minerals were introduced into the interstices of the sedimentary rocks long after they were laid down.

H. Physical Condition of Metallic Minerals.—Metalliferous components of unweathered igneous rocks are completely locked up in them. They may be released through weathering or, below ground-water level, through metamorphic change. In the latter case the contribution to general ore-formation is necessarily small. In the former, there can be but little doubt that, through the breaking down of igneous rock-masses under exposure to the elements, the total aggregate of metals liberated must be very great.

3. *Magmatic Origin of the Metals.*

A. Association of Ore-Deposits with Cooled Magmas.—That there is a general diffusion of ore-materials through molten magmas is indicated in a number of ways. Minerals containing the heavy metals are found to be present in rocks which have manifestly solidified from a molten state. Under conditions of high temperature and great pressure, the metallic compounds seem to have existed mainly in volatile form. In the cooling of magmatic masses, the metals are largely expelled, as is shown by the character of volcanic emanations. This effect offers an explanation of the fact why so many metalliferous deposits are found at the contact of igneous masses with the rocks through which they break.

The metals dissolved in magmas have a varied history. A part is often segregated through differentiation before the solidification of the mass. Another portion is expelled in gase-

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[45] *Missouri Geological Survey*, vol. vii., p. 479 (1894).

[46] *Iowa Geological Survey*, vol. x., p. 567 (1900).

ous form as the magma cools, and, forming soluble combinations, readily passes into the general ground-water circulation. Still another considerable fraction remains behind in scattered crystals or diffused in other minerals, forming in the first instance accessory rock-constituents, which are among the first minerals to crystallize out, and, in the second case, a component or included part of the ferro-magnesian silicates, which are usually products of a subsequent generation of crystals.

While, then, it is quite manifest that the heavy metals are widely disseminated through molten magmas and in the igneous rocks resulting from their refrigeration, it is necessary in the consideration of the derivation of ores to make a clear distinction between ore-materials from the two sources. In magmas the metals are easily segregated or liberated. After magmas become crystallized into rocks the accessory constituents and the other metalliferous minerals are securely locked up.

B. Deep-Seated Origin of Metals.—The fact that the average density of the earth (5.6 according to Sterneck [47]) is more than twice that (about 2.5) of the rocks forming the lithosphere, while according to Laplace's law a density of 10.74 should exist at the center of the earth, makes the deep-seated region apparently, as Posepny remarks, "the peculiar home of the heavy metals."

C. Magmatic Waters in Ore-Deposition.—Reasoning from these premises and the phenomena displayed by cooling extravasations from volcanoes, Suess [48] suggests that ascending ground-waters are often, and perhaps usually, magmatic in character, and bring to the surface of the earth their burdens of metallic salts from the barysphere. DeLaunay [49] more recently adopts this explanation and applies it generally. Lindgren [50] suggests it for the origin of certain gold-bearing quartz-veins of Victoria and California. For some Alaskan ore-deposits Spencer [51] also ascribes a magmatic origin of the vein-filling waters.

The principle is, no doubt, of far wider application than is commonly ascribed to it.

Suess's view is strongly supported by the results of the most

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47 *Mittheilungen des kaiserlich-koenigliches militär-geographisches Institut.*

48 *Engineering and Mining Journal*, vol. lxxvi., No. 1, p. 8; No. 2, p. 52 (July 4, 11, 1903).

49 *Contribution à l'étude des gîtes métalifères*, p. 6 (1897).

50 *Engineering and Mining Journal*, vol. lxxix., No. 10, p. 460 (Mar. 9, 1905).

51 *Trans.*, xxxvi., 364 (1906).

recent investigations concerning the condition of the earth's interior. Considering, in accordance with the latest advances in the physical sciences, the behavior of gaseous, liquid, and solid masses under conditions of high temperature and great pressure, Arrhenius[52] estimates that in a gaseous state about one-half of the globe should consist of iron and other metals mingled with it. He says:

"If the rocks at the earth's surface have a density half that of the globe as a whole, and if the density continues to hold good for the magma that arises from the melting of these rocks, we must conceive the existence of a much denser substance in the earth's interior. On various grounds, such as the preponderance of iron in nature, both in meteorites and in the sun, and the phenomena of terrestrial magnetism, it may be inferred that this substance is metallic iron. In consequence of its greater density this iron will naturally lie deeper than the rock-magma, and, on account of the high temperature, must exist in a gaseous condition. Somewhere about a half of the planet, therefore, should consequently consist of iron, and of the other metals mingled with it in smaller proportions. The semi-diameter of this gaseous iron-sphere will thus include about 80 per cent. of the earth's semi-diameter. Then will come about 15 per cent. of the gaseous rock-magma; next to it the liquid rock-magma for a thickness of about 4 per cent. of the terrestrial semi-diameter; and lastly the solid crust, for which not more than about 1 per cent. may be claimed."

D. Ore-Materials of Volcanic Emanations.—The recent observations on the gaseous emanations of volcanic eruptions, especially those of Vesuvius, Santorin, Krakatoa, Cotopaxi, and other well-known volcanoes, clearly show that chlorine, fluorine, and boron exist in large quantities, and, next to steam, are the most abundant of the vapors present. The bright yellow iron chloride, and the emerald-green copper chloride, besides less conspicuous metallic compounds, are of common occurrence in and near the vents of the volcanoes. These, as well as other facts, seem to indicate very strongly that at high temperatures it is the gases of this class which act chiefly upon the metals in magmas. Such compounds of the metals, volatile at high temperatures, accumulate at or near the surface of the eruptive or intrusive masses as they cool. All of them are easily soluble and readily pass into the underground-water circulation.

Notable ore-deposits containing the heavy metals in the form of chlorides and fluorides are infrequent. As Clarke[53] has recently remarked,

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[52] *Geologiska Föreningens i. Stockholm Förhandlingar*, vol. xxii., p. 404 (1900).

[53] *Bulletin No.* 330, *U. S. Geological Survey*, p. 549 (1908).

". . . . in other forms chlorine and fluorine have acted as primary agents in bringing about their concentration, and water tends to hydrolyse the salts thus formed, other solutions react with them, and quite different compounds are precipitated. In the case of tin the oxide is commonly produced; the other metals tend to appear as sulphides. Chlorine and fluorine act only as temporary carriers of the metals, and when their work is done they enter into other combinations. Fluorine remains in a gangue-mineral, fluorspar; the chlorine returns into circulation as a soluble alkaline chloride—that is, as common salt"

These are the simplest cases.

E. Exportation of Metals from Magmas.—Modern petrology has incidentally given us insight into the processes and the conditions attending the segregation of ore-materials around igneous intrusions. When, through mountain-making movements, lines of weakness are developed in the earth's crust, and molten magma is forced upward, the rocks which it traverses are not only fractured and brecciated, but more or less extensively displaced. Often the molten rock gathers into large bodies or lakes before it solidifies.

When the molten body of magma begins to cool, it shrinks, producing cracks and crevices in the hardened margins. Gases from the interior are extruded, much in the manner often noticed in a pot of liquid slag from the smeltery. Into the crevices, both of the solidified portion of the igneous mass and of the surrounding rocks into which the intrusion has taken place, mineralized water with its dissolved burden finds access. Part of this heated water is magmatic in character, part is quarry-water, and part is meteoric water that has found its way into the deep circulation.

With the mingling of water from different sources, there arises an intricate succession of chemical reactions. Extensive precipitation, especially of metallic compounds, takes place in the surrounding and relatively cold country-rock. Soluble chlorides and fluorides of the metals that are volatile at high temperatures, and which naturally accumulate at the edges of the intrusions, pass into the general ground-water circulation, in which they migrate until they enter into other combinations. Most true contact-ores seem to be formed in this way.

F. Ores of Magmatic Differentiation.—The most recent investigations concerning the magmatic differentiation of rocks show that three distinct sorts have to be taken into account—namely,

static differentiation, taking place in the depths of the earth; differentiation through cooling during ascent towards the surface of the earth, according to the principle of Soret; and crystalline differentiation. These three forms of magmatic differentiation are especially emphasized by the eminent Russian authority, Loewinson-Lessing,[54] in his great work on the eruptive rocks of the Central Caucausus.

A magma in which heavy metals tend to segregate is doubtless exceptionally rich in those components. It is well known, says Clarke,[55]

> ". . . . that several sulphides exist as magmatic minerals and that they are more abundant in some places than in others, varying in this respect just as the feldspars do. In other words, the magmatic constituents are not uniformly distributed throughout the crust of the earth. A magma, then, with more than the average proportion of sulphides, rises to the surface of the earth and cools progressively. In so doing, some segregation of sulphides must take place, and they become thereby concentrated at the margin of the cooling mass. The product of concentration may itself appear as a large and distinct ore-body, like the Norwegian pyrrhotites, or it may be relatively trivial; but in either case the first step has been taken."

The magnetite-deposits of Jacupiranga and Ipanema in the province of Sao Paulo, Brazil, lately described by Derby,[56] appear to be of similar nature.

Of the metalliferous components which are retained as accessory rock-forming minerals in the mass of a crystallized magma, the sulphides of iron and copper and the oxides of iron and tin are the most abundant. When the subject of ore-materials in the igneous rocks shall have been thoroughly investigated, it is likely not only that many other metalliferous minerals than those now known will be detected, but that the views now dominant will undergo radical change. The magmatic origin of ore-deposits is only one phase of the general problem, and perhaps not the most important one.

4. *Meteoritic Derivation of the Metals.*

A. Planetesimal Hypothesis.—It follows from the premises of the planetesimal theory of the earth's origin, as recently and

[54] *Congrès Géologique International*, St. Petersburg, Mémoires, p. 53 (1897).

[55] *Bulletin No.* 330, *U. S. Geological Survey*, p. 550 (1908).

[56] *Quarterly Journal of the Geological Society of London*, vol. xlvii., Part 2, p. 251 (May 1, 1891).

specifically set forth by Chamberlain,[57] that there is falling upon the surface of our globe a constant rain of rock-forming material coming directly from extra-terrestrial sources. That such a perpetual shower really takes place seems now to be fully demonstrated by many facts. That it is an important general source of ore-materials appears also sufficiently probable to warrant more serious consideration than it has yet received.

The meteoritic theory is not new. So long ago as 1848 Meyer[58] presented a well-supported hypothesis of an origin of the planetary and stellar bodies through meteoric agglomeration, which has received, since that time, the support of many distinguished authorities.

That portion of the stellar dust which falls upon the land mingles immediately, almost unnoticed, with the soil. That part which falls into the sea goes to form the characteristic bottom-muds of the ocean. Whether falling on land or sea, the stellar-dust particles, on account of their high specific gravity and their prevailingly metallic character, tend sooner or later to sink deeper and deeper beneath the lighter material on which they are deposited; but local physical conditions at or near the surface may affect the usual direction of this migration so as to bring together the materials of ore-bodies.

Upon ultimate analysis, the meteoritic hypothesis is not so wholly novel, and so radically distinct from the nebular hypothesis of Laplace, as some of its advocates would have us believe. G. H. Darwin has shown[59] that the meteoric swarm is dynamically analogous to a gas; and in reality the laws of gases strictly apply to it.

At the present time the planetesimal hypothesis has special attraction in its bearing upon the ultimate origin of the ores. It explains many aspects of ore-genesis that have long remained enigmatical. It does away with the sweeping claim that ores owe their formation entirely to volcanic activities; and it suggests the vadose zone as the seat of the principal segregation of ore-materials generally.

B. Number of Meteorites.—We know something of the larger meteorites that have fallen to the earth; and our prevailing no-

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[57] *Carnegie Institute Yearbook*, No. 3, p. 208 (1905).

[58] *Beiträge zur Mechanik des Himmels*, p. 157 (1848).

[59] *Philosophical Transactions of the Royal Society of London*, vol. clxxx., pp. 1 to 69 (1889).

tions of extra-terrestrial materials are largely confined to these occurrences. It is, however, the constant and almost inappreciable shower of cosmic dust and particles falling upon the earth's surface that is of greatest consequence as a possible source of ore-supply.

The magnitude and persistency of the stellar-dust shower ordinarily escapes notice. It is rendered visible in various ways. Hailstones are frequently found containing small particles of presumably meteoric iron. By the melting of snow in the arctic regions fine metallic particles composed mainly of iron, nickel, cobalt, copper, etc., are obtained.

The banded appearance of arctic glaciers is well known. Its main cause seems to be layers of fine dust and minute rock-fragments. Nordenskjold,[60] in particular, calls attention to the banded appearance of certain arctic snow-fields, in which the dark zones were found to be due to minute black grains, most of which were metallic in character. Chamberlain,[61] in presenting some fine photographic views of the fronts of Bryant, Krakokla, and other Greenland glaciers, specially emphasizes the conspicuous banded appearance. While he incidentally states that the dark particles are "mainly terrestrial," he gives no data upon which he bases his conclusion; and he leaves it to be inferred that he regards at least a part of the material as perhaps meteoritic in character. The myriads of dust-wells which the same author[62] describes in the surface of the great Igloodahomyne glacier seem to have like significance.

The great abundance of chrondres in the abysmal deposits which cover the floor of the ocean is especially noted by Murray and Renard[63] in the reports of the *Challenger* expedition. These masses are largely composed of basic minerals, closely related to the earthly substance known as bronzite; and, with small doubt, appear to be of cosmic origin.

Some conception of the reality and importance of the heavenly swarm which is constantly reaching us may be gained when it is remembered how frequent and numerous are meteoritic falls. In each 24 hours there are, according to Young,[64]

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[60] *Comptes rendus de l'Académie des Sciences*, vol. lxxvii., p. 463 (1873).

[61] *Journal of Geology*, vol. iii., No. 5, p. 568 (July-Aug., 1895).

[62] *Ibid.*, No. 2, p. 215 (Feb.-Mar., 1895).

[63] *Narrative of the Cruise of H. M. S. Challenger*, vol. ii., p. 809 (1885).

[64] *Astronomy*, p. 472.

no less than from 15,000,000 to 20,000,000 of meteorites entering the earth's atmosphere.

C. Abundance of Meteorites in Desert-Regions.—The frequency of meteoric irons and meteoric stones in the arid regions of the globe, and especially on the high, dry plateaus, is particularly significant in this connection. While such falls are probably not more common in those districts than elsewhere, the peculiar climatic conditions tend to give them prominence. The clear air, the cloudless skies, and the high altitudes contrast sharply with the thick atmosphere and the prevailingly cloud-covered firmament of the sea-coast of humid lands. In the high, dry regions, the frequency of meteoric manifestations immediately arouses the wonder of the sojourner from cloudy lands. The constant stream of light-paths across the heavens reminds one, every night in the year, of the November meteoric showers of other parts of the world.

Moreover, a dry climate seems to prevent rapid rock-decay. There is practically no such phenomenon as chemical decomposition of the rocks as it is known in the moister regions of the globe. The breaking-down of rock-masses near the surface takes place mainly by means of strictly mechanical disintegration. Meteoric iron remains for years on the surface of the desert without notable oxidation.

Again, the meteorites in such regions, instead of being immediately lost to view among vegetation, covered by earth, and subject to rapid chemical decay, as in humid countries, are left exposed on the surface through the constant removal by the winds of the lighter soils of arid lands.[65] This cause affects, of course, all the larger rock-fragments, of whatever origin. The pebble mosaics which cover large tracts of arid plain, described by Blake,[66] by Tolman,[67] and by Keyes,[68] amply attest the extent of this remarkable phenomenon.

For peculiar reasons, meteoric masses are not easily recognizable in the pebble-pavements. The majority of desert rocks are susceptible to notable discoloration and wind-polishing, which imparts to them a burnt and fused appearance.

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[65] *Bulletin of the Geological Society of America*, vol. xix., p. 73 (1907).
[66] *Trans.*, xxxiv., 161 (1904).
[67] *Journal of Geology*, vol. xvii., p. 149 (1909).
[68] *Ibid.*, p. 74.

Travelers in the desert are prone to ascribe this characteristic of the rocks to volcanic action; and it is one of the features of such regions which always attracts their attention. For example, in describing the general impressions gained in crossing the broad desert tract in New Mexico, known as the Jornada del Muerto, Wallace[69] says: "The portion I speak of appears to have served its time, worn out, been dispeopled and forgotten; the grass is low and mossy, with a perishing look—the shrubs, soap-weed, and bony cactus writhing like some grisly skeleton; the very stones are like the scoria of a furnace." Until they are broken in two, the darkened rock-fragments give little suggestion of their real lithologic character.

The more basic rock-masses and larger rock-fragments which strew the ground throughout the arid regions are almost invariably coated with a black iron and manganese film, which, highly polished by the wind-blown sands and dusts, gives every appearance of fusion. The aspect thus produced is not unlike that of the fused surface of meteorites falling in moist lands. Among such dark, lacquered rock-fragments it is with the greatest difficulty that true meteorites can be distinguished.

That they do occur abundantly, nevertheless, is well shown by the rock-collections displayed at every cattle-ranch.

A notable instance of the exceptional frequency of meteoric iron in the desert-regions, and one which has recently attracted wide attention, is that of the Canyon Diablo falls in eastern Arizona, first brought to notice by Foote.[70] Twenty miles east of that isolated and majestic volcanic pile known as the San Francisco mountains, in the midst of the vast level plain forming the general surface of the High Plateau, is a low mound, locally called Coon Butte. The center of the low elevation is occupied by a crater-like depression about 1,000 ft. across. In the vicinity of this hill such large amounts of meteoric iron have been collected from time to time as to give rise to the notion that the crater was produced by an enormous meteorite striking the earth at this point,[71] the impact causing the frag-

[69] *Land of the Pueblos*, p. 140 (New York, 1888).

[70] *American Journal of Science*, Third Series, vol. xlii., No. 251, p. 413 (Nov., 1891).

[71] G. P. Merrill: *Smithsonian Miscellaneous Collections*, vol. l., p. 461 (1908).

ments to scatter in all directions. As a matter of fact, meteoric irons are not more abundant at Coon Butte than they are in other parts of the dry country, or, probably, in the desert tracts of the earth generally. At Coon Butte, a company has been led into expending thousands of dollars in sinking shafts and drilling for the supposed great heavenly iron body deeply inclosed in the bowels of the earth. The central depression itself is in reality a true volcanic crater of the explosive type;[72] but the accidental finding of many pieces of meteoric iron within it and about it has stimulated the imagination of observers, who have given undue weight to these occurrences as indicative of the origin of the crater. The occurrence of such meteorites, instead of being special and novel, is general and wide-spread in desert-regions. It is to the arid tracts of the globe that we must look for the greatest extension of our knowledge concerning meteoritic materials.

It is to the desert-regions likewise that we must turn for information regarding the character of the rain of stellar dust. The remarkable prevalency of black-sand grains in the desert soils deserves the notice which it has generally escaped. On the vast high plains of the dry Mexican plateau, metallic particles occur abundantly in the soil, miles away from the mountains and from outcrops of igneous rocks. The plains are so level, the distance from the mountains so great, and the rainfall so scanty, as to preclude the easy transportation of these heavy particles by means of water, while their high specific gravity must prevent their movement by means of the winds. Yet after the severe rain-showers which occur at rare intervals, when little rills traverse the surface in all directions, considerable quantities of the "iron-sands" accumulate along the paths of the moving waters. A thorough chemical investigation of the composition of these sands would be highly instructive. The common black sands of placers appear to be in the main totally distinct; and their origin is usually traceable to decomposing igneous rocks. The metallic sand-particles of the desert soils necessarily long resist decay. Should these particles prove to be of meteoritic origin, the fact would tend to make such estimates of annual meteoritic augmentation to the

[72] *Bulletin of the Geological Society of America*, vol. xviii., p. 721 (1906).

earth's volume as those given by Chamberlin and Salisbury[73] ridiculously inadequate. As it is, their figures must be vastly too low.

D. Petrologic Character of Meteoric Falls.—When the petrologic features of the larger stony masses called meteorites are examined in the same way as the igneous rocks of our globe, suggestive relationships are at once established. Among the common terrestrial rocks of igneous origin there are usually recognized four main groups: the acidic, the intermediate or neutral, the basic, and the ultra-basic. Among earth-rocks, those of the last-mentioned class are quite rare; but in the case of stony meteorites the rock-species distinguished are very largely ultra-basic. Some of these mineralogic aggregates correspond, it is true, to some of the most basic of the terrestrial series; but the cosmical series begins with the earthly basic class and continues through the ultra-basic series, of which limburgite and peridotite are the chief terrestrial examples, to yet unnamed series in which the metals form a larger and larger proportion.

In 1871, Meunier[74] recognized among the meteorites no less than 50 distinct lithologic types, of most of which he later[75] described the microscopical characters and reported in them a wide range of metallic elements.

E. Metals in Meteorites.—The metals found in the meteorites include nearly all of those occurring in the common ores. Gold and silver are the only conspicuous metals which do not yet appear to be found abundantly in meteorites. There are, however, good grounds why these two metals have not been reported; and other reasons why certain other metals seemingly occur only sparingly; so that the apparent absence of these elements in the composition of known meteorites does not preclude their derivation from this source.

As is well known, the comparative abundance of elements appears to be different in the terrestrial and meteoric rocks. It is suggested by Farrington,[76] however, that there is good reason for believing this difference to be apparent rather than

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[73] *Geology*, ed. 2, vol. i., p. 381 (1905).

[74] *Geologie des Meteorites: Moniteur scientifique Quesneville* (Feb. 15, 1871).

[75] *Bulletin de la société d'histoire naturalle d'Autun*, vol. xvi. (1893); vol. xvii. (1895).

[76] *Journal of Geology*, vol. ix., No. 5, p. 394 (July–Aug., 1901).

real. Only the crust of the earth is considered; and the analysis of meteoric material does not often show the true proportion of stony matter.

F. Incorporation of Meteoritic Materials in Ores.—On the theory of meteoritic agglomeration, the original and often the immediate source of ore-materials cannot be in nature so largely magmatic as it is vadose. Qualified in some ways and strengthened in others, the general arguments of Forchhammer, Sandberger, Winslow, Van Hise, and Bain assume a new interest and an added value. The main shortcoming, if such it really be, is merely in ascribing a sole or principal origin of the ore-materials to rock-weathering, when a somewhat broader interpretation of the facts seems necessary.

It is not difficult to fancy the manner in which metallic substances of meteoritic origin may become incorporated with the ore-materials generally. After reaching the surface of the earth, both cosmic dust and the larger meteorites must mingle with the soil, more or less quickly oxidize, and enter, by means of the circulating ground-water or otherwise, the deep-seated zone, in the same way as any of the heavier mineral particles liberated from the surface-rocks through decomposition are supposed to do. The processes involved are essentially the same as for the changes and movements of rock-forming ore-materials. The distinction to be made is that, instead of the ore-materials being derived from the breaking-down of the rocks of the lithosphere, a very large proportion is regarded as coming from extra-terrestrial sources.

Although there is probably no such universal sea of ground-water as that pictured by Van Hise, there is yet no reason for not believing that surface-water readily penetrates to the deep region, even to the zone of rock-flowage. The lithosphere thus represents merely the flotsam and jetsam of the globe, through which the heavier materials may migrate, generally inward as individual particles, but occasionally or spasmodically outward, in connection with volcanic flow.

In the course of the inward migration of ore-materials temporary ore-bodies are often localized, in the vadose zone chiefly. How much of these materials are of recent extra-terrestrial origin and what proportion is the product of rock-decay, is at present difficult to estimate. The meteoritic contribution has

received as yet insufficient attention. That it may be much more important than has been suspected hitherto, is clearly shown by recent observations in desert-regions. That this is the main source of vadose ore-materials now seems not unlikely. It is probable that most of the diffused metallic content of the sedimentary rocks is in reality immediately derived from meteoritic sources; for its derivation from the country-rock of mining-districts, especially in tracts far removed from volcanic activity, has never been a very satisfactory explanation.

III. Recapitulation.

Inferentially, the supplies of metalliferous materials derived from meteoritic sources probably equal, if they do not actually exceed, those liberated by the secular decay of rock-masses. It is also a serious question whether, of the productive ore-bodies of the world, the majority are not in the so-called vadose zone. A glance at the vast descriptive literature on the mines of the world seems to give support to this statement.

During the past few years the consideration of ore-deposits has been vastly complicated; and the time seems opportune to attempt to view in something of their true perspective both of the main theories of ore-genesis. The claims of the one school need not necessarily antagonize those of the other. It appears now that they are mutually supplementary.

The broad relations of ore-migration into the interior of the earth seem to have far greater significance with reference to the origin of ore-bodies than is furnished in the case of any local outward movement from cooling magmatic masses. As regards the localization of ore-materials into workable ore-bodies, this view greatly emphasizes also the importance of geologic structure.

Of the four principal ultimate sources of ores, that first mentioned, from sea-water, may be entirely neglected, for one reason, if not for many others—namely, because such a process must be one of general diffusion, rather than of concentration. There seems to be no known instance in which ore-bodies have been segregated through this means.

In connection with the other three sources the conditions are complex. In the release of ore-materials from rocks containing them, or from space, and their subsequent migration in solu-

tion through the ground-water circulation, the main tendency is also towards diffusion and not concentration. Accidental geologic structures and conditions retard free and universal migration, and permit local precipitation and temporary segregation of metallic minerals. Meteoritic materials mingle with those derived from rock-decay and then obey the same laws.

In the order of their industrial importance, the several ultimate sources of ore-materials appear to be: (1) meteoritic accumulation; (2) lithologic decomposition; and (3) magmatic expulsion.

The Genesis of the Leadville Ore-Deposits.

BY MAX BOEHMER, DENVER, COLO.

(Pittsburg Meeting, March, 1910.)

AFTER 30 years of development and after an output of $350,000,000 in value of gold, silver, lead, zinc, and copper, there has not been published a satisfactory explanation of the origin of the immense deposits of the Leadville district.

The original examination and survey of Leadville by members of the U. S. Geological Survey was a most magnificent piece of work, but their attempt to explain the origin of the ores was based upon the then-accepted theory of lateral secretion, originated by Sandberger, which theory was afterwards disproved in many cases by further development in numerous districts.

This theory contended that the ore in the veins had been leached by surface-waters from the eruptive rocks in the immediate vicinity, and it had its origin in the fact that the neighboring eruptive rocks all contained, to an appreciable degree, the several metals found in the ore-deposits.

But the practical miner, reasoning in a simple manner from the condition of things as he found them underground, soon discovered the fallacy of this proposition. He reasoned that, in the case of the Leadville deposits, this theory could not be true, because, if the ores were leached from the overlying eruptive rocks by surface-waters, they would as a matter of necessity be controlled by the action of gravity, and the main bodies

would collect in natural depressions of the underlying limestone, and at all points where such depressions existed. But this was not in accord with the channels of ore as he found them; and hence the theory of lateral secretion in this district was never accepted. Another objection to it consisted in the fact that large areas within the ore-bearing district contained no ores whatever, although their formation was the same, and the same limestones were covered by the same thick sheet of eruptive rocks. The experts of the Geological Survey themselves soon saw that their original theory was not satisfactory, and it is expected that their later examination and survey of the district will throw a new and correct light upon the subject. But we do not know when this report will appear; and, meanwhile, others, who have given some thought to the subject, may be permitted to offer their ideas in the light of the latest developments.

That the eruptive rocks play a very important rôle in the genesis of most ore-deposits is now universally acknowledged. It seems also well established that the fluid magma of the eruptive rocks at great depth originally contained the metals which are found in the veins; that these metals were dissolved and segregated from the magma, then held in solution in depth, until pathways towards the surface were opened, when, under the influence of heat and pressure, the solutions rose in these pathways and deposited their burden wherever the conditions, either physical or chemical, were favorable.

In Leadville, it was discovered that the later eruptives, and not the earlier white porphyry, were the rocks most intimately associated with the position and trend of the ore-bodies; but mining developments did not give sufficient evidence upon which to base a plausible theory of the origin of the ores, since these later eruptives, as a rule, were found in the shape of sheets or irregular tongues and bosses. A few vertical dikes had been found, but they carried no ore on their flanks, nor was there a single ore-bearing fissure in evidence on Fryer, Carbonate, Rock, or Iron hills.

Breece hill also, in its earlier development, showed similar conditions. It is only the work done in the last five years on this latter hill which has at last, in my judgment, at least, solved the question and furnished the evidence upon which the gene-

sis of the Leadville ore-deposits may be safely formulated. And this evidence is so plain that all should agree in the interpretation of the facts as lately disclosed.

The developments now show that the Leadville deposits have their origin in depth through the channels of fissure-veins and dikes of eruptive rock. Numerous fissure-veins, carrying valuable ores and generally associated with nearly vertical dikes, have been opened along the entire extent of Breece hill, and beyond it for a distance of 3 miles. Their course is in all instances approximately N–S., and their dip is from 70° to 80° west or east. These fissures send out to the left and right large channels of ore into the adjacent stratified rocks, especially at the limestone horizon below the cover of the main sheet of white porphyry. This mineralization in the limestone extends at least a mile from the fissures, and the latter carry values down into the underlying granite.

In a number of cases, the fissure which admitted the dike-magma and later the vein-waters faulted the country; and it is important in exploitation to distinguish these pre-mineral faults from the later post-mineral faults which created the present topography of the country.

The veins accompanying these dikes generally lie along the foot-wall or the hanging-wall, but occasionally also within the body of the dike.

Frequently these dikes show no valuable ores associated with them, but a number carry along their flanks a mass of nearly clean quartz with scattered pyrites, which is probably not a segregation, but an actual low-grade deposit, made by the vein-waters in the form of silicified country-rock.

After the uplift of the region, the surface-waters caused a secondary rearrangement of ores and of values within the ore-bodies to a surprising degree; and extensive migration of the metals from upper to lower horizons took place in the stratified rocks, invading even the upper portion of the hard and glassy, lower quartzite, where rich, secondary ores are found in vugs and water-courses below much larger masses of ore of lower grade in the limestones directly above.

These fissures and dikes exist only in the easterly portion of the Leadville district, within the so-called gold-belt, where the deposits carry gold in addition to the silver, lead, and other

metals—no gold being found in the westerly portion of the district.

It seems reasonable to assert that the proximity of the gold-belt to the original fissures and dikes explains the existence of gold in the veins of that particular area. Gold would naturally be precipitated first; and thus it did not find its way, except in a solitary instance, to the westerly portion of the district.

Fig. 1, an ideal east and west section of Breece hill, fully illustrates the main features of these later developments.

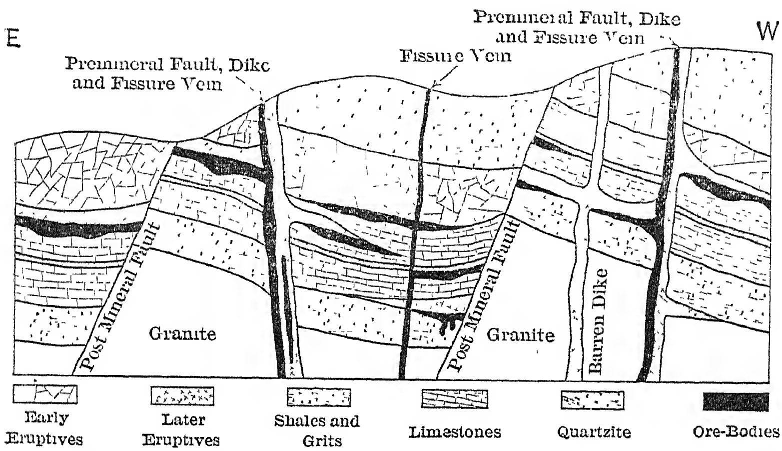


FIG. 1.—IDEAL SECTION OF BREECE HILL, LEADVILLE, COLO.

In this connection, it occurs to me that other districts, such as the lead- and zinc-deposits of Missouri, have been interpreted wrongly in their genesis, since the discoveries in Leadville demonstrate that the original source of the ores may be situated to the distance of a mile or more away from the ore-bodies found and developed in the mines.

I am sure that the most recent discoveries in Leadville throw much new light upon the difficult problem of the genesis of ore-bodies everywhere, and I hope that the U. S. Geological Survey will soon publish the result of its latest investigations in this district.

Exploration of Certain Iron-Ore and Coal-Deposits in the State of Oaxaca, Mexico.

BY J. L. W. BIRKINBINE, PHILADELPHIA, PA.

(Pittsburg Meeting, March, 1910.)

INTRODUCTION.

THIS paper is a discussion of a part of the mineral wealth of the States of Oaxaca and Puebla, Mexico. It does not refer to the precious metals, some mines of which, in these States, are said to have been worked before the advent of the Spanish *conquistadores* in the sixteenth century, as my purpose is to invite attention to the exploration of deposits of coal and iron-ores in the Mixteca region. The prominence given to coal is warranted by the great effect of the scarcity of fuel upon the industrial development of Mexico. Under the progressive administration of President Porfirio Diaz, the Republic has made wonderful advances in the past three decades; and this progress would have been greater but for the limitations imposed by expensive fuel.

In the spring of 1906 I was detailed by the Birkinbine Engineering Offices, of Philadelphia, to accompany A. B. Adams, of New York, in a reconnoissance of reported deposits of iron-ores in the State of Oaxaca. The result of this expedition was the formation, in 1907, of the Oaxaca Iron & Coal Co., of which Mr. Adams became, and has remained, President and General Manager, and I have spent the past three years as Chief Engineer of this company in Mexico, principally in western Oaxaca, where field-headquarters were established.

GEOGRAPHY AND POPULATION.

Although the geographical and topographical features of the Republic of Mexico are well known, few realize that the majority of its population is domiciled in the States which immediately surround the capital city of Mexico, including the Federal District, Mexico, Tlaxcala, Morelos, Puebla,

Oaxaca, Vera Cruz, Hidalgo, San Luis Potosi, Queretaro, Guanajuato, Aguascalientes, Jalisco, Colima, and Michoacan. These States cover 177,600 sq. miles of the grand total of 767,300 sq. miles. The census of 1900 gave these 15 States a population of more than 9,485,000, out of 13,606,000 for the entire Republic. In other words, 23 per cent. of the area of the Republic contains 70 per cent. of its population. With the possible exception of the portion of the State of Vera Cruz adjacent to Gulf ports, the present price of coal in this region ranges from $7 to $15 a ton, on board cars at railroad terminals. (The values are given in gold. Mexican currency is taken as worth 50 per cent. of gold-value.) Within the region numerous mining and industrial centers are seeking economical power; and coal, crude oils, and gas (utilized in gas-engines), as well as hydro-electric installations, have received attention and encouragement. Mexico City being the center of population, industry, and commerce in the Republic, the prices of various commodities at that place are given in this paper. The metric ton of 2,204.6 lb. is used, unless some other unit is stated.

Fuel in Mexico.

Wood and charcoal, exclusively used as fuel in the past, are still largely relied upon, even in important cities; but the rapid destruction of forests adjacent to lines of transportation has greatly augmented the cost of vegetable fuel throughout the Republic, thus encouraging the use of coal, which is now generally burned under boilers, and has been utilized lately in gas-producers, the gas thus generated being supplied to heating-furnaces and to internal-combustion motors for power. Some industries are applying waste charcoal-breeze in special producers to operate gas-engines; and numerous locomotives are burning crude native petroleum.

The most densely populated part of Mexico is distant from present fuel-supplies. The wood along the railroads entering the valley of Mexico has been depleted; and the only domestic coal now available is mined more than 800 miles to the north; while foreign coal must be transported from Gulf ports, a distance of 264 miles by minimum haul, overcoming an elevation of 8,400 ft. before descending into the valley of Mexico, the altitude of which is 7,300 ft., and necessitating a charge of

$4.75 per ton for freight between Vera Cruz and Mexico. Foreign coal, therefore, costs from $10 to $11 per ton in Mexico City, and about 25 cents less in Puebla. Anthracite coal from the United States and "Crown fuel" briquettes, imported from Great Britain, command from 50 cents to $1 per ton more than bituminous coal.

In the northern part of Coahuila, extensive coal-deposits have been developed, but they are more than 800 miles from the City of Mexico, and 6,000 ft. below it in altitude; and the railroad freight-charge averages about $4.50 per ton, making the domestic coal sell in Mexico City at from $8.50 to $10 per ton. The Coahuila coal is high in ash (from 15 to 18 per cent. by average analyses of imperfectly-washed coal); indeed, coal from this field carrying as much as 25 per cent. of ash has been sold in Mexico City. These deposits were described by Edwin Ludlow in a paper presented at the Institute meeting in Mexico, November, 1901.[1]

Coal has been found also in the State of Sonora, still further from Mexico City. These deposits were described by Prof. E. T. Dumble[2] before the Institute in 1899.

Petroleum, which is obtained along the Gulf coast, commands about the same price per ton as Crown fuel-briquettes or Pennsylvania anthracite, being handicapped by the same transportation-charges to Mexico City. Lately the wells have been damaged by the intrusion of salt water, which, while not interfering with the character of the refined products, affects the value of crude oil as fuel, and thus restricts consumption, especially on the railroads.

The Mixteca Region.

Near the central portion of the Republic, between the parallels of latitude 15° and 18° 30′ north, and between the meridians of longitude 20° to 21° 15′ west of Washington, is the Mixteca region, embracing about the western third of the State of Oaxaca and the southwestern portion of the State of

[1] The Coal-Fields of Las Esperanzas, Coahuila, Mexico, *Trans.*, xxxii., 140 to 156 (1902).

[2] Notes on the Geology of Sonora, Mexico, *Trans.*, xxix., 122 to 152; and Natural Coke of the Santa Clara Coal-Field, Sonora, Mexico, *Trans.*, xxix., 546 to 549 (1899).

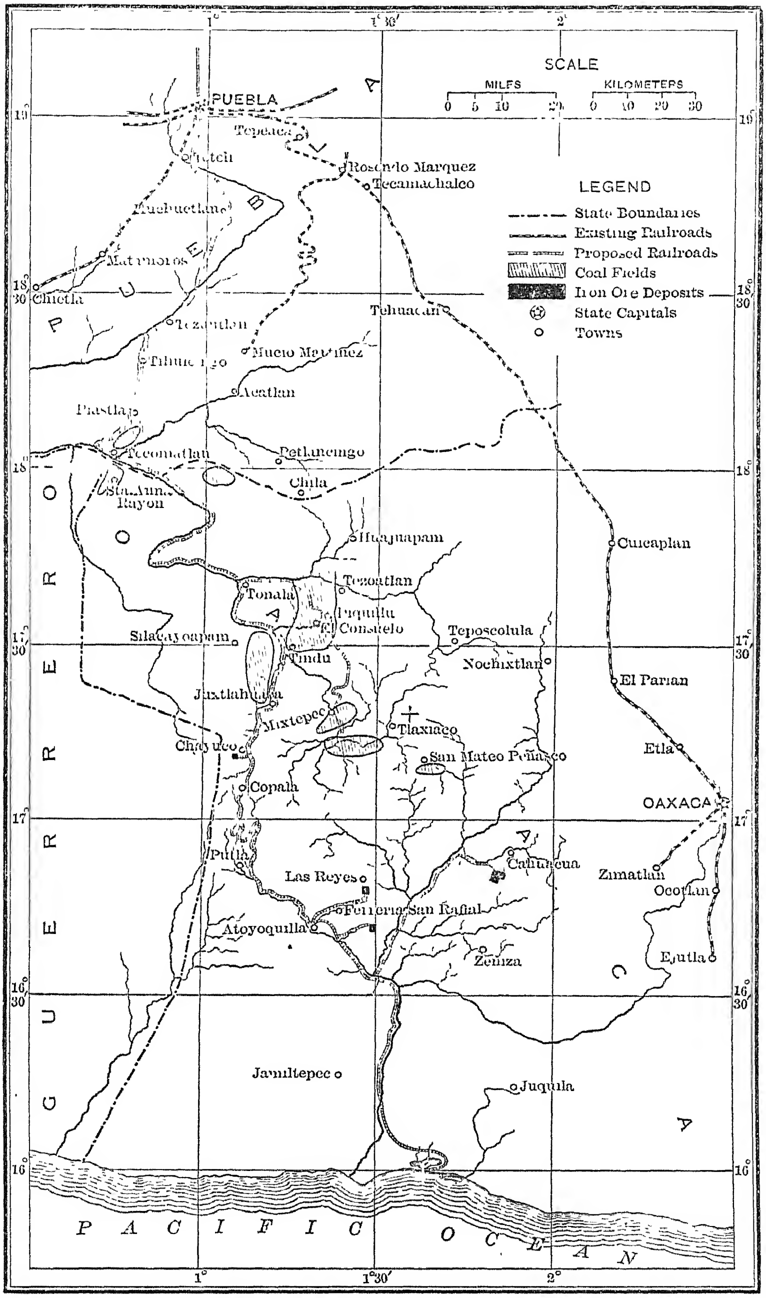


FIG. 1.—MAP OF THE MIXTECA COUNTRY, STATES OF OAXACA AND PUEBLA, MEXICO, SHOWING COAL-BASINS, IRON-ORE DEPOSITS, AND EXISTING AND PROPOSED RAILROADS.

Puebla. The country is mountainous, extending from sea-level at the Pacific ocean to 10,000 ft. above, the average altitude being more than 5,000 feet.

Its ancient history, like that of many other parts of Mexico, is unknown; but numerous ruins and peculiar languages or customs, which still prevail, indicate that the original Mixtecas had reached a comparatively high plane of civilization. According to common legend, supported by the opinion of some archæologists who have lately visited this portion of Mexico, the Mixteca Indians antedate the Aztecs, and it is possible that they accompanied the Toltecs when these entered Mexico from the north. In some parts of Oaxaca there are colonies of Indians known as "*triques*," believed to have been originally communities of prisoners gathered from the numerous campaigns of the Mixtecas against other tribes. What can be learned of the history of the Mixtecas indicates that they retained their independence until after the arrival of the Spaniards in 1525; and it is commonly asserted that the Mixtecas and the Tlaxcalans were the only two independent tribes that withstood the conquest of the war-like Aztecs, to whom even their former allies, the Zapotecas, were finally forced to yield.

The Mixteca region is divided topographically into two parts, known as the Mixteca Arriba and the Mixteca Baja; and inhabited by separate clans, exhibiting still marked differences in language.

The Mixteca Indian is usually peaceful and industrious, and develops, with very little training, into a skilled workman with a tendency to be busy before the working-day begins and to continue long after hours in order to complete the task once begun. He has a happy disposition and works for very small wages, so long as he may remain in his own part of the country, to which he is strongly attached. Although always craving for an opportunity to visit Mexico City, he is just as anxious to return to his own land, as soon as he has seen some of the modern wonders of the capital.

In our reconnoissances in the spring of 1906, we were probably the first American party to make an extended visit to this part of Mexico. For weeks at a time we traveled on horseback through this unmapped portion of the Republic with perfect security, receiving everywhere not only courtesy but

sincere hospitality. Doubtless many of the attentions we received were due to letters and instructions sent by General Diaz (who, like the other great executive of Mexico, Benito Juarez, was a Oaxacan), and by the present Governor of the State, Emilio Pimentel. Yet, even *peones*, who had no way of knowing that we were bearers of official letters, were spontaneous in their courtesy and assistance, showing their desire to aid us in every way. In three years of nearly continuous residence among these people, I have found them always courteous, reliable, and honest. So much has been published concerning the indolence, inefficiency, and dishonesty of the Mexican *peon*, that the above statement, based upon long and intimate association, is offered as an offset to the generally-accepted opinion.

Labor.

The Mixteca Indians furnish an ample supply of labor, and, with little training, become skilled in the duties assigned to them. The following rates of wages prevail throughout the work:

Day-laborers, unskilled, and speaking only the Mixteca dialects, receive from 9 to 12.5 cents per day of from 10 to 12 hours.

Day-laborers, unskilled, but speaking Spanish, and also mine-muckers, receive 12.5 to 18 cents per day.

Gang-foremen, blacksmiths, and carpenters, 31 cents per day.

General foremen and special office- and laboratory-man, 50 cents a day.

They appear to be born mechanics, soon learning to fire the drill-boilers, to operate the hoisting-engine, and to make, out of old pieces of available scrap, any small parts necessary for repairs. They can be trusted, if not too severely tempted, and, if treated with kindness and firmness, they are extremely loyal. For packing heavy loads they are invaluable and indefatigable. Like all other Spanish-American countries, Mexico has innumerable feast-days; but hitherto our men have been perfectly willing, in cases of necessity, to let these go by. Their chief weakness is a love for alcoholic drinks, especially *aguardiente* (which is practically pure alcohol); and their favorite time for its consumption is on Sundays and holidays.

Surveys.

The exploratory concession granted to the company embraced nearly 11,000 sq. miles, the existing cartography of which was either meager or inaccurate. It was therefore necessary to survey this area. There is a government triangulation-system (known as the triangulation upon the meridian 98° west of Greenwich) embracing much of this region. The stations are from 20 to 70 miles apart, and we were consequently obliged to make a secondary triangulation at essential points. Two base-lines were measured, one in the vicinity of Tlaxiaco, 1,691.785 m., and the other, 1,126.12 m. long, in the vicinity of Tezoatlan. The first, or Tlaxiaco, system of triangulation embraced an area of 360 sq. km. (more than 140 sq. miles), with several high peaks at greater distances (located by observations from two points). One of the government triangulation-stations formed part of this system. The Tezoatlan system included 600 sq. km. (more than 230 sq. miles). From the secondary triangulation-stations, third systems were extended to cover smaller areas, embracing the coal-fields at Mixtepec and at Mina Consuelo. In the vicinity of Mina Consuelo and of Tlaxiaco, the meridian was determined, and permanent monuments were erected.

It may be of interest to detail the method of obtaining base-line measurements, in a country where a satisfactory straight line on ground, which was level or had a uniform slope, could not be secured, and the system carried out in the triangulation. The base-line was determined with a standardized steel-tape, 100 m. long, supported at intervals of 5 m., measurements being taken when the temperatures of the ground and of the air were practically uniform, with a strain of 15 lb. on the tape. The supporting-stakes were on uniform grade-lines, and each end of a grade-line was considered as an intermediate station. The difference in elevation between these ends of grade-lines was subsequently determined by means of a level. Two sets of measurements with the tape and two sets of levels were run over the base-line by different observers, and the results were calculated. When these checked within 5 mm. the mean was adopted as the final measurement. From the ends of the base-line the triangulation was made with a Gurley light mountain-transit, graded to read to 30″, the angles being repeated 10

times, and the total being divided by 10 gave the reading within 3″. Where any angle or measurement of a series differed from the mean by 30″ or more, a new series of readings was taken. The light mountain-transit was adopted, although having the disadvantage of a small limb, because its ease of transportation and stability in adjustment made it superior to the larger transits for the use to which it was put. All transits were equipped with full vertical circles and two with Segmuller solar attachments for determining the meridian at distant points by observations on the sun; also one with a repeating vertical arc, in order that leveling by vertical angles and determination of latitudes could be accurately made.

In addition to the above-mentioned triangulation-systems, 400 acres have been covered by accurate topographic survey, and sketch-topography covering 1,250 sq. miles has been made. Most of this work has been done by Assistant Engineer H. N. Roberts, who deserves credit for accuracy, thoroughness, and close attention to details. Practically all the concession of 11,000 sq. miles, as well as several thousand square miles in the State of Puebla, has also been reconnoitered on horse-back or on foot.

Equipment.

As it required from 3 to 5 days for mail to pass from the headquarters at Tlaxiaco, and from 5 to 7 days from the field-headquarters of Mina Consuelo, to Mexico City, while freight and express consumed from 10 to 30 days in transit, it was necessary to equip the corps liberally, including a miniature field drug-store for accidents or sickness (which, fortunately, has been little used in the past three years). Drafting-rooms were established at the two above-mentioned points for plotting maps and geological data, the designing of head-frames and small ventilation-systems for some of the drifts, etc. A field-laboratory, first established at Tlaxiaco, but later moved to Mina Consuelo, was equipped to make chemical analyses of iron-ores and coals, and provided with apparatus for certain physical tests, including a Parr calorimeter and a petrographical microscope. Several crude pieces of apparatus were constructed for experimental tests in sizing or washing coal. Two "C" Sullivan diamond-drills, with water-tube boilers, pumps, drill-rods, core-barrels, casing, stand-pipe, and necessary tools

and extra parts, were transported to the field-headquarters by mules. Hand-drills, sledges, tools, blacksmith and carpenter outfits, portable forges, provisions, and camp-equipment were also provided. At Mina Consuelo it was necessary to erect all buildings, which at first were of the native style, bamboo tied with *reatas* (*i. e.*, ropes made from palm-leaves) and a roof made from the palm; but these were afterwards superseded by a log-cabin, in which the logs were held together by pins of wood, with split-shingle roof. Dwelling-houses, laboratory, and dining-room were built of logs and split shingles, as well as the head-frame at the shaft, store-house, stable, and blacksmith-shop.

Since the only hand-tools available in the field of our explorations were of ancient design (with which, however, the natives did excellent but slow work), the equipment supplied included what would be a fair stock for a small hardware-store in the United States.

Our assistants came from the United States, England, and Canada, but the packing and other labor was done by a liberal force of natives, whose wages, however, did not make the work expensive.

The reconnoitering parties were equipped with horses, saddles, *serapes* (native blankets), and *ponchos* for each member; and each party had also an aneroid barometer, a Brunton compass, a Maignen filter, one or two prospecting-picks, medicine cases, note-books, and pencils. Exploratory parties were also supplied with camp- and cooking-equipment.

Geology.

The coal-deposits in the Mixteca region must have been known for more than half a century. There is an old drawing, bearing the title, *Croquis de la Area Carbonifera de Tlaxiaco descurbierta por Jose Vincente Comacho en 1850.* (Sketch of the Tlaxiaco Carboniferous Area discovered by Jose Vincente Comacho in 1850), upon which are shown several drifts and outcrops; but no work, except a few short drifts scattered over a large area, and a trench near Mina Consuelo, had been done in the region prior to 1907.

The first geological study of the Mixteca region, of which records can be found, was made by Inginiero Santiago Ramirez, who examined, in 1881, some coal-outcrops near the boundary

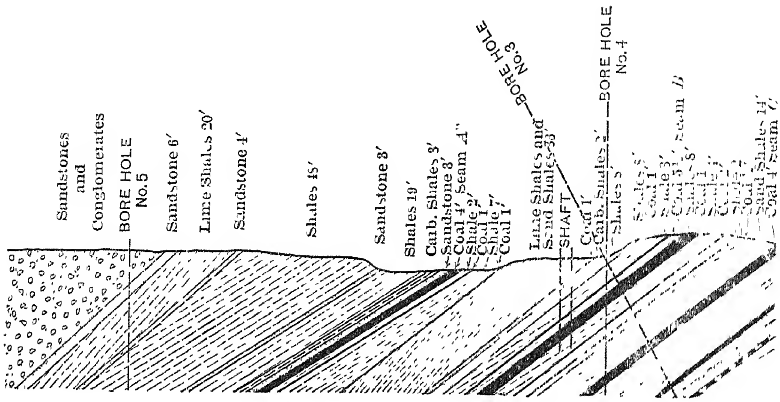
between Oaxaca and Puebla. In the same decade, Messrs. Felix and Linke made geological studies in the vicinity of Tlaxiaco; and in 1887–1888 Sr. Jose G. Aguilera, now Director of the Instituto Geologico Nacionale, examined the vicinity of Tezoatlan and the northern part of the State of Oaxaca. The activity of the Oaxaca Iron & Coal Co. attracted to this undeveloped field the interest of the National Geological Institute, which, in the fall of 1908, sent an engineer to visit the Mixteca region. He collected for the Institute considerable geological data and numerous fossils, and examined some coal-outcrops, which, however, he regarded as possessing no importance, since he was able to find but few samples that carried less than 18 per cent. of ash, which he considered to be a maximum for useful coal. In the spring of 1909, the Director himself accompanied me in a brief tour of inspection to some of the deposits; and somewhat later commissioned another party, composed of Prof. G. R. Wieland and Inginiero Bonilla, to visit the Mixteca region. They spent several months in the field; but the work of these geological parties was directed rather to the correlation of the various strata, to petrographical determinations, and to the collection of fossils, than to economic geology.

Geology of Western Oaxaca.

In the territory under discussion the general geology may be described as follows:

The lowest formation is the Archaic, consisting of gneiss, mica-schists, and mica-slates. On this are superposed small areas of Jura-Trias, while over larger areas appears the Cretaceous formation. The pre-Cretaceous Mesozoic rocks generally consist of shales (varying greatly in composition), coarse and fine sandstones, and conglomerates, also some quartzites. The Cretaceous is represented mostly by massive limestone, although in some parts slates and calcareous sandstones are found. Above the Mesozoic formations occur in some places the Tertiary red sandstones and conglomerates, and in other places "*caliche*," which is either of Tertiary or Quartenary age.

Throughout these various formations, although more predominant near the junction of the Archaic and the upper sedimentary rocks, large areas are covered by Tertiary intrusives



A

(andesites, basalts, rhyolites, etc.), while in some localities lava-flows and masses of obsidian are found. The Mesozoic formations of the region are extremely interesting to palæontologists, on account of the large number of fossils which they contain. The Jura-Triassic carries a great variety of mollusca and plant-

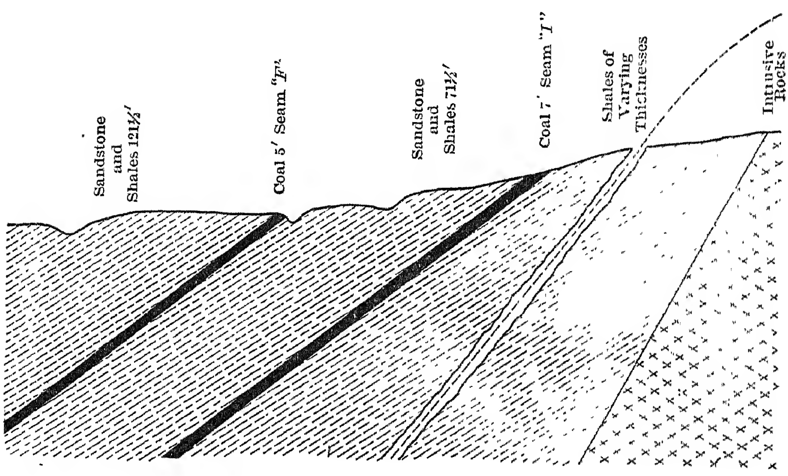


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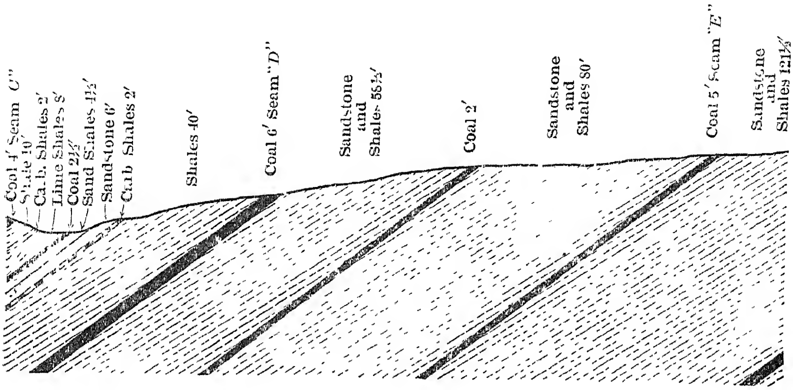
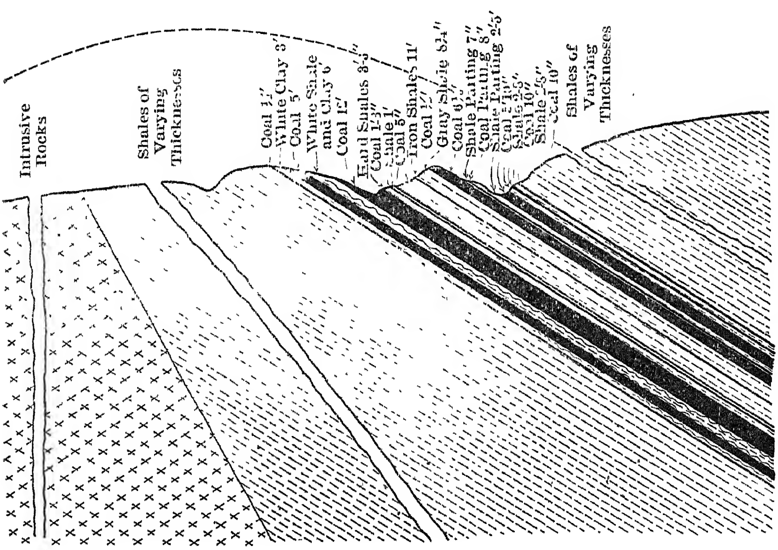
Fig. 3 (A, B, C, D).—Geological Section, Vicinity of Mina Consuelo, Bore-Holes



B

forms. The mollusca include several forms of *trigonia* and *stefonigero;* while among the plant-forms cycads are predominant, although there is beyond doubt a great variety of other forms. Professor Wieland, in his paper entitled The Williamsonias of the Mixteca Alta, says: "I am of the opinion that



D

OAXACA, MEXICO, AT RIGHT ANGLES TO THE STRIKE-LINE AND THROUGH NOS. 3 AND 4.

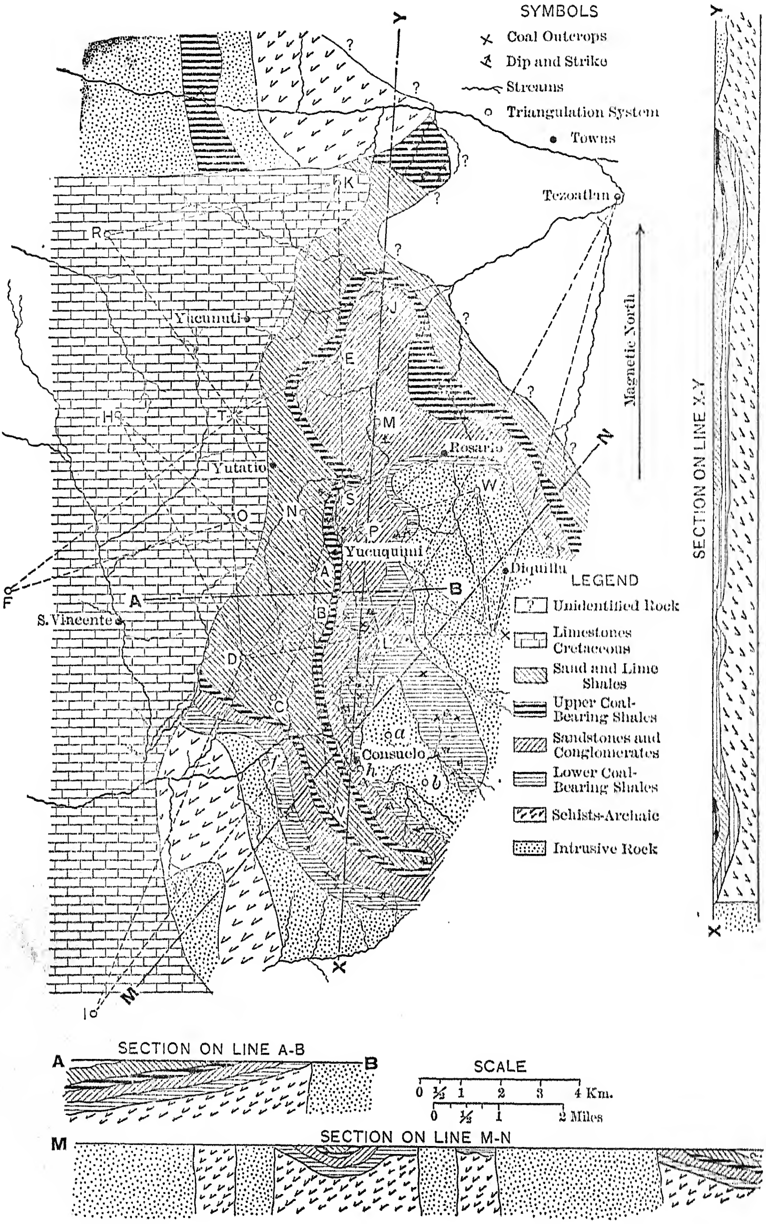


FIG. 2.—GEOLOGICAL MAP AND SECTIONS, VICINITY OF MINA CONSUELO, OAXACA, MEXICO.

the Mixteca Alta is one of the most promising and accessible regions for the student of fossil plants yet discovered."[3]

The various formations of the pre-Cretaceous Mesozoic have not been correlated, being grouped under a general term as Jura-Trias; but the thick Cretaceous limestone is sufficient as a geological horizon for field purposes; and an intermediate horizon has been used, consisting of layers of black oyster-shells, and called by the members of the corps "the black shell-rock."

The Tezoatlan coal-field has been more closely examined than any of the others, and the following section of it may be regarded as typical:

Downward Section of Tezoatlan Coal-Field.

Cretaceous limestones,	massive and of great thickness.
Calcareous and arenaceous shales, including the "black shell-rock,"	about 500 ft.
Upper coal-bearing shale,	110 ft.
Sandstones and conglomerates,	800 ft.
Lower coal-bearing shales,	at least 800 ft.
Intrusive or archaic rocks.	

The upper coal-bearing shales have not been examined, except superficially, since the lower shales appeared to have greater value. Several sections have been made of various portions of the lower coal-bearing shales, which vary in thickness according to where they are cut off by the intrusive rock. The following is offered as representative:

Section below the 800 *Feet of Sandstones and Conglomerates Noted in Preceding Section.*

	Ft.	In.
Sandstones,	6	0
Calcareous shale,	20	0
Sandstones,	4	0
Ferruginous, calcareous, and arenaceous shales,	48	0
Sandstone,	3	0
Ferruginous and arenaceous shales,	19	0
Carboniferous shales,	3	0
Sandstone,	3	0
Coal. Seam A,	4	0
Shale parting,	2	0
Coal,	1	0
Blue shale,	7	0
Coal,	1	0

[3] *The Botanical Gazette,* vol. xlviii., No. 6, p. 427, *et seq.*

	Ft.	In
Arenaceous and calcareous shales,	33	0
Coal,	1	0
Carboniferous shales,	2	0
Blue shales.	8	0
Arenaceous shales,	8	0
Coal,	1	0
Shale,	3	0
Coal. Seam B,	5	6
Shale,	8	0
Coal,	1	0
Shale,	1	0
Coal. Seam C,	2	0
Shale parting,	2	0
Coal,	0	6
Arenaceous shales,	14	0
Coal,	4	0
Shale,	10	0
Coal,	2	0
Calcareous shales,	8	0
Coal,	2	6
Arenaceous shales,	4	6
Sandstones,	6	0
Carboniferous shales,	2	0
Shales,	40	0
Coal. Seam D,	6	0
Sandstones and shales,	58	6
Coal,	2	0
Sandstones and shales,	80	0
Coal. Seam E,	5	0
Sandstones and shales,	121	6
Coal. Seam F,	5	0
Sandstones and shales,	71	6
Coal. Seam I,	7	0
Sandstones and shales,	?	
Coal,	0	10
Shale,	2	8
Coal,	0	10
Shale,	2	8
Upper split of coal seam No. 3,	5	10
Shale parting,	2	5
Coal parting,	0	8
Shale parting,	0	7
Lower split of coal-seam No. 3,	6	4
Gray shale,	8	3
Coal,	0	6
Iron shales,	11	0
Coal,	0	5
Shale,	1	0
Coal,	1	3
Hard shales,	3	3
Coal. Seam 2,	12	0
White shale and clay,	6	0

	Ft	In.
Coal. *Seam* 1,	5	0
White clay,	3	0
Coal,	0	6
White and iron-shales of varying thickness down to intrusive rock.		

This section shows a total of 83 ft. 2 in. of coal, in which 15 seams over 2 ft. in thickness aggregate 72 ft. 8 in., and 9 of these, exceeding 3 ft., give an aggregate thickness of 64 ft. 8 in.

The intrusive rocks cut these formations at various points, but, in the coal-fields proper, the nearest they come to the coal-seams is (excepting one or two places) about 100 ft. below coal-seam *K*. Coal-seam *K* is not designated in the section. On account of the undetermined rocks below seam *K* and above seam 3, the nomenclature of the seams was changed at this point.

The formations are faulted and folded, but not so much as would be expected. The dip is generally between 30° and 50° west at Mina Consuelo, and the same amount to the east at the opposite side of the basin. In places, the coal lies horizontal in small areas, while the faults, with the exception of quite a large one on practically the axis of the synclinal (where there has been a displacement of nearly 1,000 ft.), are unusually small, although numerous. Three faults have been found, of 200, 120, and 55 ft. displacement, respectively, while there are many others which vary from a few tenths of an inch to a foot or more. The general strike in the vicinity of Consuelo is N.-S., but in approaching the town of San Juan Diquillu it swings around to E.-W.

Coal-Fields and Character of Coal.

The field-headquarters were located in Tlaxiaco, which, although more than 80 miles by horse-trail from the nearest railroad-station, has a population of 8,000, a small water-supply, electric light, and a large number of stores. The relative locations of the various fields will be referred to this place.

In the Penasco field, 10 miles SE. of Tlaxiaco, a high-grade coal in small deposits was found by an exploring-party. A sample from the Junuzma mine gave, upon analysis, moisture, 9.45; volatile matter, 25.85; fixed carbon, 60.45; and ash, 4.25 per cent.

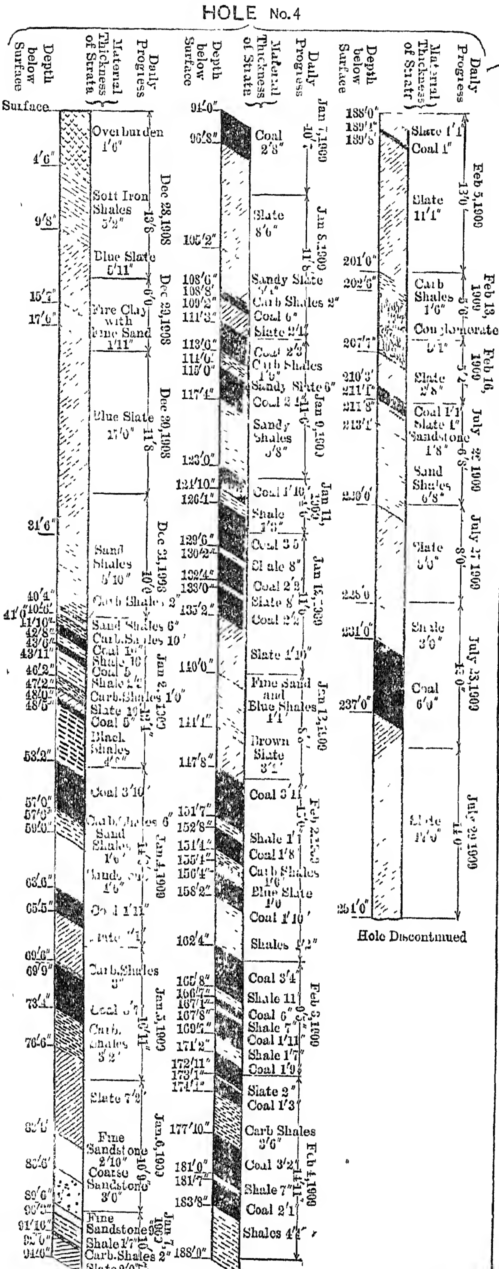


FIG. 4.—RECORD OF DIAMOND-DRILLING AT MINA CONSUELO, OAXACA, MEXICO.

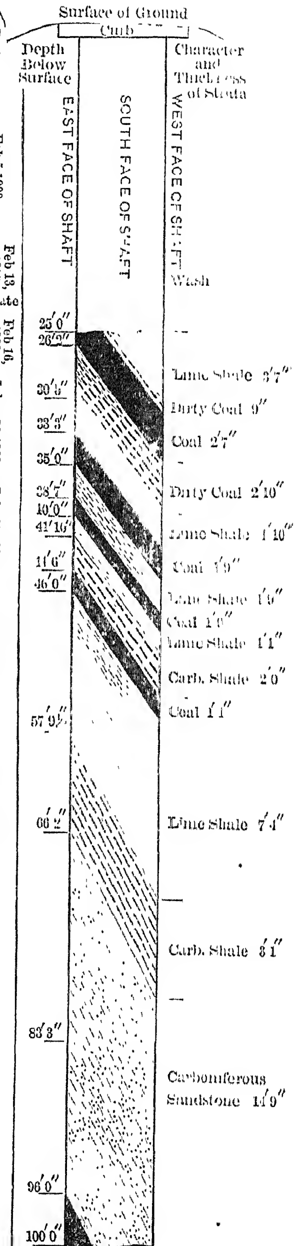


FIG. 5.—SECTION OF SHAFT A ON AN EAST AND WEST LINE, MINA CONSUELO, OAXACA, MEXICO.

The Tepejilla field, about 20 miles NW. of Tlaxiaco, covers a small area; and the coal found there is high in ash and apparently small in quantity.

The Juxtlahuaca field includes outcrops in the vicinity of the towns of Juxtlahuaca and Silacayoapam, where preliminary investigation showed the coal to be non-coking and high in ash. Samples showed on analysis the following results:

	Juxtlahuaca. Per Cent.	Silacayoapam. Per Cent.
Moisture, }	5.9	9.27
Volatile matter, }		27.36
Fixed carbon,	42.0	39.93
Ash,	52.1	23.44
Analyzed by	O. I. & C. Co.	Instituto Geologico.

The Tecomatlan field, embracing the outcrops in the vicinity of Tecomatlan, in the State of Puebla, and Santa Ana Rayon, Oaxaca, is 75 miles NW. of Tlaxiaco. The coal here is soft and pulverulent and showed on analysis:

	Per Cent	Per Cent.
Moisture,	1.8	2.0
Volatile matter,	17.1	18.3
Fixed carbon,	57.1	48.7
Ash,	24.0	31.0

The Tlaxiaco field, the Mixtepec field, 12 miles west of Tlaxiaco, and the Tezoatlan field, 25 to 30 miles NW. of Tlaxiaco, are those in which most development-work has been done, and will be described in detail under separate headings.

In addition to the above localities, coal is reported as occurring in four or five other places; but samples showed it to be of inferior quality.

Tlaxiaco Field.

Since the company first acquired control of this field, its work was started here, and was greatly facilitated by the progressiveness of the people of Tlaxiaco, who, following the example of Sr. Salvador Bolanos Cacho, the *Jefe Politico* of the district, not only secured samples of iron-ore and coal from distant points, but aided us with their advice and in every possible way made our work pleasant and comfortable. I take this opportunity to express my appreciation of the uniform courtesy and kindness shown us by every one within

the boundaries of the State, from the Governor, Sr. Emilio Pimentel, down to the *Presidente* of the smallest town.

The Tlaxiaco field was sub-divided into three tracts, the Villaverde, the Stein, and the Rio Tlaxiaco. The work done upon the Villaverde and Stein tracts consisted in mapping and uncovering 14 outcrops and securing samples, which show the coal to vary greatly in composition, much of it appearing to be too poor for commercial use. Analysis from one of these coals gave the following results:

	Per Cent.
Moisture,	2.07
Volatile matter,	16.77
Fixed carbon,	52.69
Ash,	28.47

The work on the Rio Tlaxiaco tract consisted of eight drifts, which had a total length of 900 ft., including cross-cuts, and exposed seams varying from 10 in. to 6 ft. 0 in. in thickness. Most of these seams are very dirty and show the effects of considerable faulting, the seams consisting of flakes of coal and slate. The following are the analyses of some of the better seams:

Seam Number	Thickness of Seam. Feet.	Moisture. Per Cent.	Volatile Matter. Per Cent.	Fixed Carbon. Per Cent.	Ash. Per Cent.
4	2	1.8	18.1	42.1	38.0
4–A	3	0.7	15.1	46.0	38.2
5	4	6.6	19.2	29.7	44.5
6	6	1.6	16.4	53.0	29.0

Mixtepec Field.

Upon the discovery of better coal at Mixtepec, the work was transferred to this locality and a large number of drifts were driven, the longest exceeding 1,100 ft. in length, which showed that the seam was 25 ft. thick and extended over a large area. Numerous samples were taken, an average of the seam showing:

	Per Cent.
Moisture,	1.3
Volatile matter,	16.2
Fixed carbon,	67.5
Ash,	13.0

Besides this seam, known as the Esperanza, there are three others, designated as Fabrica, Soledad, and Southern. The Fabrica seam, 6 ft. thick, gave:

	Per Cent.
Moisture,	1.24
Volatile matter,	16.21
Fixed carbon,	60.23
Ash,	22.32

The Soledad seam, 5 ft. thick, showed:

	Per Cent
Moisture,	1.06
Volatile matter,	14.03
Fixed carbon,	66.69
Ash,	16.22

The Southern seam, 3 ft. thick, gave:

	Per Cent.
Moisture,	13.69
Volatile matter,	28.78
Fixed carbon,	25.38
Ash,	32.15

The Southern seam, which appears to be rather a lignite than a true coal, is located about 4 miles from the Mixtepec field proper.

All of the above analyses represent the "run-of-mine," the large pieces of slate only being removed. As the Mixtepec coal is quite soft, some crude tests showed that the ash could be reduced to one-half the original content by sizing on revolving screens, while washing or jigging would make a still greater reduction.

As the percentages of ash appeared high, two samples were taken and tested in a calorimeter to determine their fuel-value. The dirty coal, carrying 29.38 per cent. of ash, yielded 11,400 B.t.u., while a clean picked sample, containing 3.85 per cent. of ash, gave 15,900 B.t.u.

As the evidence of the value of the coal-fields appeared to increase greatly upon examination, it was decided to purchase a diamond-drill; and, the nearest point to the railroad being the Tezoatlan field, the drill was sent there. Later a second drill was erected at the same place, and the entire force was moved to Mina Consuelo.

Tezoatlan Field.

For the past 18 months all the development-work of the company has been confined to the Tezoatlan field, and here the work has reached its highest development, although still in progress. Thirty-five drifts have been driven into the coal in order to show the continuity of the coal-seams along the outcrop, while seven diamond-drill holes and a shaft have been sunk to determine its extent in depth. More than 71 sq. miles have been covered by a geological survey; and the data thus collected have been mapped, while detailed geological and topographical surveys have been completed on 350 acres, and are now in progress on 1,000 additional acres.

As shown in the geological section, the coal-seams 3 ft. or more thick in this locality have a total true thickness of 64 ft. 2 in., although in part of the field the intrusive rocks have cut out the lower 29 ft. 2 in. of the seams, leaving available 35 ft. of coal. As the average dip is 30° or more, these true thicknesses will be equivalent to vertical thicknesses of 74 and 42 ft., respectively, and would yield, according to the rule of thumb (that 1 ft. vertical thickness gives a yield of 1,200 tons of coal per acre), 88,800 and 50,400 tons per acre, respectively. The upper 35 ft. of coal has been traced over an area of 3,000 acres, while the total thickness of 64 ft. has been traced for a distance of 1.25 miles, though the work has not yet reached a stage permitting the determination of the area underlain by the total thickness of seams. The coal may be called an anthracite, being hard and dense and burning without smoke, a typical analysis showing:

	Per Cent.
Moisture,	1.0
Volatile matter,	5.5
Fixed carbon,	73.5
Ash,	20.0
Sulphur,	0.06

11,500 B.t.u.

This analysis represents the coal when mined and picked; the "run-of-mine," unpicked, carrying about 25 per cent. of ash.

The Tezoatlan coal-field is a large basin, extending in a general N-S. direction, the distance between the eastern and west-

ern outcrops being, near the southern end, about 2 miles, while, on the north, the western outcrop is hidden by the Cretaceous limestones, which are unconformable to the lower strata.

An interesting feature of this coal-field is, that the intrusive rocks, which are considered to be Tertiary, have had practically no effect on the coals. In some places, coal-seams are found occurring with surprising uniformity within 20 ft. of the intrusive rocks. The formations of the various strata in this vicinity are extremely interesting; and the rapid alteration of the strata (consisting of coal, shale, fine and coarse sandstones) shows that there was a constant variation of the depth of water during deposition.

Iron-Ore.

Although the preliminary reconnoissance in 1906 had for its object the investigation of certain deposits of iron-ore, with the idea of utilizing them in the manufacture of iron, using either charcoal or imported coke as fuel, the coal-deposits appeared to be of more immediate value than those of iron-ore; and therefore nearly all the work has been done on the fuel-deposits.

There is no doubt that a waiting market exists for iron- and steel-products in Mexico, as these now command high prices on account of freight charges and import duties. Pig-iron sells for $40 per ton in Mexico City, castings at $50 per ton and upward; and the duty on pig-iron is $5 per ton, while that on certain finished products is as high as $150 per ton.

The iron-ores in the State of Oaxaca are of high grade. Thirty-three samples, taken from within an area of 4 sq. miles and tested in the field-laboratory, showed an average of 60.87 per cent. of metallic iron. George C. Davis, chemist, of Philadelphia, made an analysis of a sample, closely representing the average of the Cahuacua ore, which showed Fe, 65.86; S, 0.06; and P, 0.03 per cent. The phosphorus and sulphur are low in all the iron-ores of this district, and in the deposit which has been most largely developed there are indications of large quantities of high-grade Bessemer ore. Samples from a deposit at El Carnero averaged 66 per cent. of metallic iron, and Mr. Davis made an analysis of a hand-sample, with the result: Fe, 63.20; SiO_2, 8.25; P, 0.024; and S, 0.03 per cent. In the locality known as La Ferreria, the average iron-content of the ore,

as determined in the company's laboratory, was 66.02 per cent.; and an analysis by Mr. Davis from a different sample showed Fe, 68.93; SiO_2, 2.80; P, 0.026 per cent. In the vicinity of Tlaxiaco iron-ores were found containing Fe, 51.71; SiO_2, 4.61; P, 0.026 per cent. Some iron-ore deposits examined in the State of Puebla gave Fe, from 42.40 to 67.0; SiO_2, from 1.30 to 15; CaO, from a trace to 8.80; P, from 0.004 to 0.051, and S, from 0.01 to 0.15 per cent.

These analyses are offered to show that ores collected from deposits scattered over a large area are rich in iron and low in sulphur and phosphorus. A few months' work at the Cahuacua deposit disclosed about 4,000,000 tons of iron-ore. The El Carnero ore is mainly magnetite; that of Cahuacua, mixed magnetite and hematite; that of La Ferreria, hematite and limonite; and that near Tlaxiaco, limonite. In the State of Puebla, the iron-ores are limonite and magnetite.

Transportation.

The extent of the exploratory work herein described was deemed essential by reason of the location of the coal-deposits as related to existing transportation-facilities. The railroads of Mexico, which now aggregate 14,000 miles, have in but few instances penetrated the mountainous section, and no existing line is nearer than 80 miles to the coal-fields. But, coal and iron-ores having been found in sufficient quantity to warrant the construction of railroad communications, reconnoissances of several railroad-routes have been made, which need not be discussed at this time. Notwithstanding the mountainous country traversed, practical routes were found, which would connect the present railway-system of Mexico with the coal- and iron-ore deposits of Oaxaca, and might be extended to the Pacific coast, with moderate curves and grades not exceeding 2 per cent.; the estimated construction-cost being moderate for the character of the territory traversed.

Mining-Conditions in the Belgian Congo (Congo Free State).*

BY SYDNEY H. BALL, NEW YORK, N. Y., AND MILLARD K. SHALER, LAWRENCE, KAN.

(Pittsburg Meeting, March, 1910.)

I. Introduction.

During the past 50 years the attention of mining-men has been turned to Africa, and within the past decade prospecting-expeditions sent into Central Africa have resulted in the opening-up of several mines. Having recently spent two years in the Belgian Congo for the Société Internationale Forestière et Minière du Congo, it is our purpose to outline briefly the mining-conditions in that part of Central Africa. Before it became, in 1908, a Belgian colony, this region was known as the Congo Free State.

II. Location and Area.

The Belgian Congo lies in southwestern Central Africa, west of the continental divide, as illustrated in Fig. 1. It has a coast-line of but 20 miles on the Atlantic, but widens rapidly eastward to points 5° north and 14° south of the equator. Its area is 908,000 sq. miles, which is almost one-quarter that of the United States and Alaska, or more than one and one-half times that of Alaska. Stated in other terms, its area is nearly one-third that of the United States, exclusive of Alaska and the insular possessions, or more than that of all the United States east of the Mississippi river.

III. Topography and Drainage.

From north to south, Africa is made up of three topographic elements: 1, the Atlas mountains, a highly-accentuated region

* Presented by permission of the Société Internationale Forestière et Minière du Congo.

of closely-folded rocks; 2, the Soudan and Sahara desert-regions, a more or less accidented plain in which the rocks are in the main flat-lying; and 3, the central and southern table-lands with the Abyssinian plateau, consisting of intensely-folded older and horizontal younger rocks.

The Belgian Congo, with the exception of the coastal

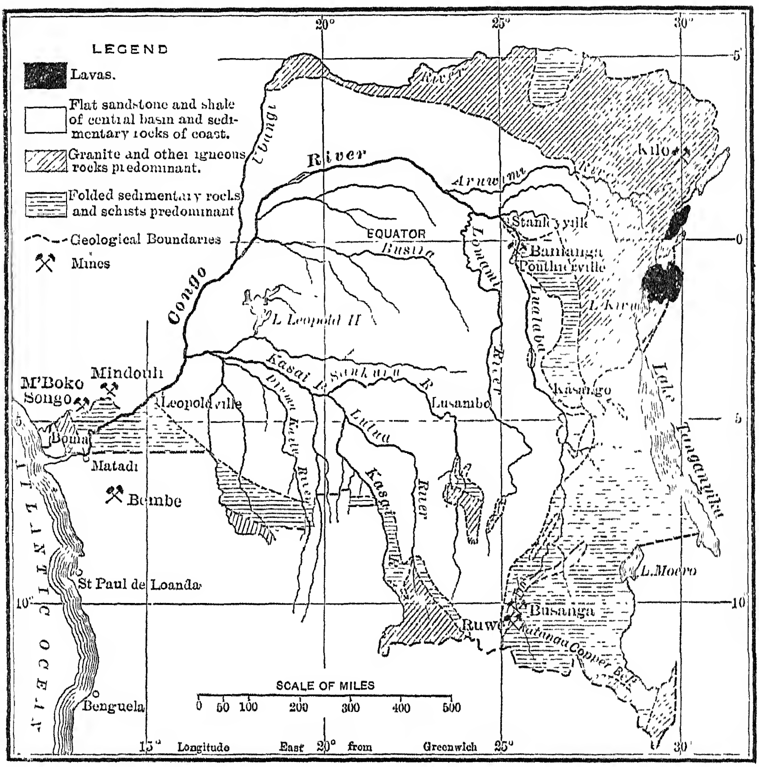


Fig. 1.—Map Showing Mining-Localities and Approximate Geological Boundaries in the Belgian Congo.

plain, lies within the plateau region and consists, from west to east, of:

1. A narrow coastal plain, which faces the Atlantic ocean, rising to

2. An ancient mountain range, now eroded to an intensely-dissected plateau, and frequently referred to as the Crystal mountains. The higher domes reach an altitude of 2,300 ft. Through the hard rocks of this plateau the mighty Congo

rushes in a series of rapids and low water-falls, the non-navigable stretch between the lower and the upper Congo rivers.

3. The great interior region, a basin whose slopes rise gently from Lake Leopold II. (altitude, 1,110 ft.). The limits of the basin to the north (altitudes from 2,500 to 4,200 ft.) and to the south (from 4,000 to 5,200 ft.) are comparatively low, but to the east rise to

4. The Eastern Frontier mountains, in height averaging about 1 mile above sea-level. Certain peaks, like the volcanic Ruwenzori (altitude, 16,800 ft.), are much higher. These mountains are in part due to folding, but largely to N–S. faulting, with which are genetically connected important volcanic phenomena.

The Congo river and its tributaries drain all of the Belgian Congo with the exception of a small area in the extreme north-east, the waters of which find their way into the Nile. With the exception of Brazil, no tropical country is blessed with such a system of water-ways as is offered by the Congo and its principal affluents, the Kasai, Sankuru, and Ubangi. Bertrand,[1] giving its length as 4,640 km., says that the Congo is the sixth longest river in the world.

Contrary to the opinion generally held regarding tropical countries, one-half of the Belgian Congo is savanna; the other half, forest. Practically all of the great equatorial forest lies between the fourth parallel south and the fourth parallel north of the equator, and is, for the most part, confined within the great horseshoe of the Congo river. Elsewhere small groves or even forests are not unusual, yet the country is typically savanna, with sparse and generally scrubby trees here and there. As the equatorial forest is approached, the streams are generally bordered by a narrow fringe of dense forest.

The Belgian Congo has a typically tropical climate, with heavy rain-fall, high humidity, and relatively high, though diurnally variable, temperature. Heavy rain-fall, or its absence, characterizes the two seasons, excepting very near the equator, where it rains more or less regularly every month in the year, and where seasons do not exist. North of the second parallel north, however, the seasons are well marked; the dry season, which

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[1] *Le Congo Belge*, p. 46 (1909.)

is considerably cooler, begins late in October and ends early in May, followed by the rainy season, which lasts six months. The seasons are, naturally, reversed south of the equator.

IV. Geology.

The geological boundaries are shown approximately on the map, Fig. 1, which gives also the mining-localities.

The coastal plain is underlain by marine sandstones, which are inclined gently towards the ocean. The sandstones, which are covered by recent alluvium at the Congo mouth, are, according to Zboinski,[2] of Tertiary and Cretaceous age.

The so-called Crystal mountains consist of older rocks, the age of which has not yet been satisfactorily determined. The strike of these rocks is NNW., and the predominant dip is east. The older rocks of the series occupy the west part of the belt, and the degree of metamorphism increases progressively from east to west. What are apparently the oldest members consist of mica-schists and sericitic quartzites, interbedded with which are chlorite- and epidote-schists, representing either basic lavas, contemporaneous with the sedimentary rocks, or very ancient intrusive bodies of igneous rocks. Of later origin are intrusive masses of granite and gabbro, now mashed and recrystallized into gneisses. Large bodies of massive granite and smaller ones of diabase also occur. To the west of these older rocks are limestones and calcareous schists, which near the ancient complex are closely folded, but are flat-lying further east, half-way between Matadi and Leopoldville. These beds are considered by E. Dupont[3] to be of Devonian age. To the east they are unconformably overlain by red sandstones and shales, the Kundulungu (Permo-Carboniferous) of Prof. Jules Cornet.[4] African geologists and engineers owe a debt of gratitude to this Belgian *savant*, as the most important contributor to Belgian Congo geologic literature, both scientific and economic.

The great interior region is covered by interbedded sandstones and shales, either flat-lying or dipping gently towards the center of the basin. The lower surface of this formation is

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[2] *Bulletin de la Société Belge de Géologie, de Paléontologie et de Hydrologie*, vol. i., pp. 36 to 41, Mémoires (1887). [3] *Ibid.*, vol. ii., pp. 47 (1888).

[4] *Ibid.*, vol. xxi., pp. 365 to 382 (1907).

undulating, and it apparently thins rapidly on the upper slopes of the basin. As the rim of the basin is approached, isolated inliers of older folded and faulted rocks begin to appear in valleys cut through the sandstone-shale blanket. Upon the rim itself these older rocks greatly predominate and are similar, in a general way, to those of the Crystal mountains.

These older rocks consist of very ancient mica-schists, quartzites, and igneous schists and gneisses, which are cut by gabbros and granites. Younger than the granites are folded sandstones, quartzites, slates, shales, and limestones. Cornet, in believing that these two series of rocks are respectively of pre-Cambrian and Palæozoic age, is without much question correct, although, unfortunately, fossils have never been found in them. In the eastern mountain-area there are also modern and Tertiary lavas, which up to date have not proved to be ore-bearing.

The main structural features on the north and south rims of the Congo basin have an E–W. trend, while the rocks of the eastern mountains strike N–S. The position of lakes Tanganyika, Kivu, Albert Edward, and Albert, and of the mountain ranges nearby, has been determined largely by faults, the trends of which are approximately parallel to the strike of the rocks.

V. Prospecting.

The flat-lying sandstones and shales of the central basin are apparently without important mineral deposits. In the older rocks, and in the gravels derived from them, bodies of ore occur, some of which are exploitable, even under the excessively high mining-costs of Central Africa. While over one-half of the Belgian Congo is thus valueless, from a mining standpoint, the rest of the country is so rich that no serious prospecting-expedition of capable men has ever returned from there empty-handed.

While trading-posts were established at the mouth of the Congo river in the early part of the nineteenth century, the opening up of the hinterland dates from Stanley's marvelous descent of the Congo in 1877, from where Nyangwe now stands. Following his discoveries, the International African Association and the International Association of the Congo were successively launched, to be replaced later by the Congo Free

State, with Leopold II., of Belgium, sovereign ruler. In November, 1908, the Belgian government took over this State as a colony. During the past 25 years no tropical country has been more rapidly explored, nor more highly developed commercially, and for the past decade prospecting has been most active.

The Belgian Congo is divided into concessions, Fig. 2, which are in the hands of more or less powerful groups of capitalists. Within these concessions, these companies have the exclusive prospecting-rights and the rights to the mines found, under conditions varying according to the articles of their charters. Throwing the region open to free, unrestricted prospecting by individuals would unquestionably cause great suffering and loss of life, since the members of improperly-equipped expeditions would undoubtedly perish, and the necessary protection could not be given by the government to all comers. To get into this country equipment, European food-stuffs, and the medicines and luxuries necessary to the white man, is an expensive undertaking.

At present, most active prospecting is being carried on by the Société Internationale Forestière et Minière du Congo, the Chemin de fer des Grands Lacs, the Comité Spécial de Haut-Katanga, and by the government itself.

VI. Mining.

Up to date the product of the Belgian Congo mines has been small. Shipments of from 70 to 80 kg. of gold per month are reported from the Kilo gold-mines, and considerable gold—10,900 oz. up to Sept. 30, 1905—has been produced from the Ruwe (Katanga) mine. Shipments, up to the present time, from the Katanga copper- and tin-deposits and from the Bamanga copper-mine, near Ponthierville, have been small and almost entirely for the purpose of milling and metallurgical testing. The copper-ore exported in 1908 was valued at $24,700. The double question of labor and transport is at present the serious obstacle to exploitation.

1. *Gold.*

This precious metal is widely distributed among the older rocks of the Belgian Congo, and is even present in small quan-

tity in certain areas within the central sandstone-shale area. Stanley states that the Arabs, at the time of his entrance into the country, washed some gold from the streams in the eastern part of the Congo basin, and Cameron writes that Zanzibarites got gold from some of the Katanga streams. Certain deposits which have been found within the past two years have not been opened up, but the following mines are being operated:

a. Kilo Gold-Mine.—This mine, Fig. 3, situated among the head-waters of the Ituri river, just west of Lake Albert, is a State property and reliable information regarding it is difficult to obtain. The predominant country-rock is granite, and free gold has been found in quartz-veins in the vicinity. The deposit, which was discovered six years ago and is now being exploited by the State, is a rich placer covering a wide area. About 25 whites, including ten prospectors, and several thousand blacks are employed. Prospecting is being constantly done in the vicinity. The monthly output, with rude sluices and very little machinery, is rumored to be from 70 to 80 kg. It is reported that a hydraulic plant is to be installed shortly.

b. Ruwe Gold-Mine.—Ruwe is situated near the Rhodesian frontier, in southeastern Belgian Congo, near the source of the Lualaba. Up to Sept. 30, 1905, the production, all obtained by sluicing the detrital deposit, was 10,900 oz. of gold. The *Tribune Congolaise* of Sept. 16, 1909, reports that the mine will shortly shut down, pending the arrival of the Cape-to-Cairo railway, as machinery is now needed.

The country-rocks are sandstones and quartzites, which dip from 12° to 35° N–E., and are cut, as is the ore-body, by faults trending NW–SE. The ore-body as developed consists of an indurated, banded sandstone, and is 1,200 ft. long, with an average width of 8 ft., but varying from 3 to 20 ft. The walls are sandstone, though at the surface, uphill from the deposit, is a band of limonite-cemented conglomerate. The values, which are patchy, average $17 per ton, and consist of gold, platinum, silver, and palladium. More or less iron, cobalt, nickel, lead, and copper are associated, the last two occurring as the vanadates, psittacinite and vanadinite and cuprodeschoizite, while pyromorphite and malachite also occur. These minerals form the "green concentrates," which are a good value-indicator. The metals platinum, palladium, and silver increase

and decrease in volume together, but gold is independent of the others. Values are rarely visible in the ore.

Immediately downhill from the outcropping ore-body, the detrital material contains nuggets, some of which are 2 in. in diameter, of gold alloyed with more or less silver. Platinum is practically lacking in this soil.

Of the two men who have described Ruwe, Studt believes that the ore was deposited by magmatic hot waters derived from what he considers an igneous dike, now represented by ferruginous clays. Buttgenbach,[5] on the contrary, believes the ore-bodies to be old beach-placers which were interbedded with the finer sediments when the sandstone series was deposited. Both agree that the gold of the detrital deposits was dissolved from the ore-body and re-deposited in the detrital material. This belief in the surface formation of the larger nuggets is based, first, on the absence, in the ore-body, of nuggets; second, on the lack of platinum in the detrital material and its presence in the ore-body; and third, on the form of some of the nuggets, which resembles, in instances, that of organic remains.

c. Other Deposits in the Katanga.—The ore of many of the Katanga copper-deposits carries a small gold- and a somewhat larger silver-content, but in rare instances only is it worth separating. Derived from the Katanga and Fungurume copper-deposits are, however, important gold-placers.

2. *Copper.*

Copper occurs widely in the older rocks of the Belgian Congo, and the Katanga Copper Range is a most important high-grade copper reserve. Of less importance are the deposits of Bamanga, near Ponthierville, and the Bembe and Mindouli copper-deposits in Angola and the French Congo, respectively.

a. The Katanga Copper-Deposits.—Mining and smelting of

[5] *Annales du Musée du Congo,* Second Series, vol. i., p. 57 to 69 (1908). *Annales de Société Géologique de Belgique,* vol. xxxiii., Mémoires, p. 65 to 67 (1905–06). *Revue Universelle des Mines, de la Métallurgie, des Travaux Publics,* etc., Fourth Series, vol. xiv., pp. 114 to 147 (1906). Farrell, J. R., Studt, etc., Tanganyika Concessions, Ltd., various reports (1900 to 1906).

copper-ores has been followed for centuries by the natives living near the Katanga copper-mines, and the majority of the mines of the Tanganyika Concessions, Ltd., are black men's mines which have been relocated by whites. Some idea of the extent of their operations is indicated by the report that at Linishia one pit is 720 ft. long, 30 ft. deep, and 480 ft. wide at one end, tapering to a point at the other. The importance of the dependent commerce in crude copper (cast into St. Andrew's crosses, worth in trade goods about $8) is shown by the fact that they reached points 400 miles away from the mines. From the crude copper, bracelets and other articles of adornment and ornamentation, and even copper wire, were made by the village coppersmiths.

At present the copper industry is unimportant among the Katanga natives. Their system of mining and smelting is as follows: Large pits are sunk in the softer parts of the ore-bodies to depths approaching 40 ft. The crumbly ore and gangue is placed in baskets and is then carried to a nearby stream, where the basket is submerged rim-deep and washed, much as one washes gravel in a gold-pan. Soon malachite alone remains, which is emptied into a pile, and the miner returns for more ore. The won malachite at nightfall is carried to the village, where it is smelted much as the iron-ore is smelted, in furnaces made usually from ant-hills. Forced draft is furnished by roughly-constructed hand-bellows.

The Katanga copper-deposits [6] are situated in southeastern Belgian Congo, about 11° S. of the equator, and not far north of the Rhodesian frontier. Up to date about 100 deposits are known, included within a belt extending 90 miles east from the Lufupa river, thence bending suddenly SE., and reaching a point 110 miles away.

Livingston, Burton, Cameron, Joseph Thomson, and Wissman all state that the natives reported rich copper-mines to

[6] Cornet, Jules, *Bulletin de la Société Belge de Géologie*, etc., vol. xvii., pp. 3 to 47, Traductions et Reproductions (1903.) Buttgenbach, H., *Annales de la Société Géologique de Belgique*, vol. xxxi., Mémoires, pp. 515 to 572 (1903–04). Buttgenbach (practically the same article), *Annales de Musée du Congo*, Second Series, Katanga, pp. 1 to 51 (1908). Farrell, J. R., and Gray, G., etc., various reports, Tanganyika Concessions, Ltd. (1900 to 1906). Farrell, J. R., *Engineering and Mining Journal*, vol. lxxxv., No. 15, pp. 747 to 753 (Apr. 11, 1908).

exist in the Katanga. Reichard[7] in 1883 was, however, the first white man to see these deposits, and he was soon followed by Capello and Ivens, Portuguese travelers, and later by the Scotch missionary, Arnot. It was, however, Professor Cornet who, in 1892, first brought back satisfactory data concerning these deposits.

In 1900 the representatives of the Congo Free State government intrusted the investigation of this mineral region to the Tanganyika Concessions, Ltd., of London. Through the able efforts of the officers of this company, particularly Messrs. Williams, George Gray, and J. R. Farrell, the American engineer, and F. E. Studt and the Belgian representative, Monsieur H. Buttgenbach, the value of the Katanga copper-region has been proved.

These deposits are situated in a plateau the surface of which has been dissected into a region of rounded hills and rather steep-sided valleys. Elevations of the higher hills and the lower valleys are respectively 5,300 and 4,000 ft. above sea-level. The copper-bearing strata are harder, as a rule, than the rocks of the surrounding country, and, in consequence, the ore-deposits are usually located on hills. These hills, in contradistinction to the surrounding timbered country, are bare of trees and shrubs, and between them and the forests grows the misuka or mahobohobo bush, which is here a most excellent copper-indicator. This de-forestation, due, unquestionably, to the presence of the deadly copper-salts in solution in the soil and subsoil, has proved a valuable aid in prospecting.

The copper-deposits are situated in quartzose strata varying from incoherent sandstones to quartzites, and in schists, slates, and limestones. According to Buttgenbach, eruptive rocks are not closely connected with the deposits. The sedimentary rocks strike generally WNW., and dip, as a rule, steeply. Sharp folding, faulting, and brecciation are common in the vicinity of the ore-deposits.

The more abundant copper-minerals of the Katanga ores, which average about 15 per cent. of copper, are malachite, chrysocolla, azurite, and melaconite, while cuprite, dioptase, native copper, and chalcopyrite are rare. Finely-divided chry-

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[7] Reisenach, Urua-Katanga, *Mittheil der Africain Gesellschaft in Deutschland*, vol. iv., No. 5, p. 303 (1885).

socolla, azurite, and malachite are found impregnating the country-rocks, and also in the form of botryoidal and mammillary masses, the three minerals being frequently closely associated with one another. Calcite layers occur in such masses, and malachite in some cavities takes the form of stalactites. Melaconite, while often associated with these carbonates and silicates, occurs also in ore intimately mixed with limonite and manganese dioxide. Beds of hematite are not rare in the vicinity of the copper-deposits, and this mineral is one of the most important gangue minerals. Besides those already mentioned, quartz is a common gangue mineral, and barytes has been found at Kambove mine No. 1. Analyses show the ore to be a siliceous one, notable for the presence of nickel and, in instances, traces of gold and silver.

The ore-bodies vary in size from comparatively small to mammoth ones, such as Kambove No. 2, reported to be 3,000 ft. long and, so far as tested, from 240 to 400 ft. wide. It is said to contain 10,000,000 tons of 15-per cent. copper-ore in each 100 ft. of sinking. These ore-bodies are remarkably alike in mineral paragenesis and in form, and probably all have a common origin. The deposits consist of lens-like masses, with their longer dimension parallel to the strike of the rocks, and in the greater number of cases the ore-body follows the dip. The ore is usually contained within rather well-defined walls. Near the center of many of the deposits, and parallel to their extension, is a sugary, sandy quartz, much fissured, in which copper carbonates and oxides occur in large masses. Away from this quartz band the country-rock is impregnated with malachite, chrysocolla, and azurite. In schists, thin sheets of these minerals occupy the closely-spaced schistosity-planes in such a way as to remind one, looking across the schistosity, of a book whose leaves are alternately green and white Where the rocks are fractured or brecciated, veins and nodules of the carbonates and oxides occur. Chalcopyrite has been found in small veins, and at the Musonai mine impregnates the sandstones. This sulphide is more or less altered to malachite, chrysocolla, and limonite.

Cornet believes these deposits to be the "iron hats" of copper sulphide bodies, as does Buttgenbach, who expects, further, that the sulphide body will consist of chalcopyrite, bornite,

hematite, and magnetite, in a vein or veins parallel to the bedding of the sedimentary rocks. The frequent presence of the medial line of quartz suggests that the original deposits were fissure-veins, with the country-rock nearby more or less impregnated with chalcopyrite. Since the Katanga has suffered notable erosion, the oxidized ore-bodies, as exposed, doubtless contain the equivalent copper-content of many feet of narrower veins which have been removed by erosion. As a result, the sulphide veins, when found, may be considerably narrower than the outcropping oxidized ores.

Lying, as it does, 11° S. of the equator and more than three-quarters of a mile above sea-level, the Katanga enjoys a climate rather favorable to white settlement. The native labor-supply is fair, timber is sufficiently plentiful for mining needs, and, while coal is lacking, electric power-plants can be readily located near the mines. The ore, being soft, is easily mined, and the deposits, being on hill-tops, can be quarried or worked by steam-shovels for a considerable time. The necessary hematite and limestone for smelting purposes are at hand, and the company engineers report that sufficient oxidized ore has been opened up to meet all demands for many years.

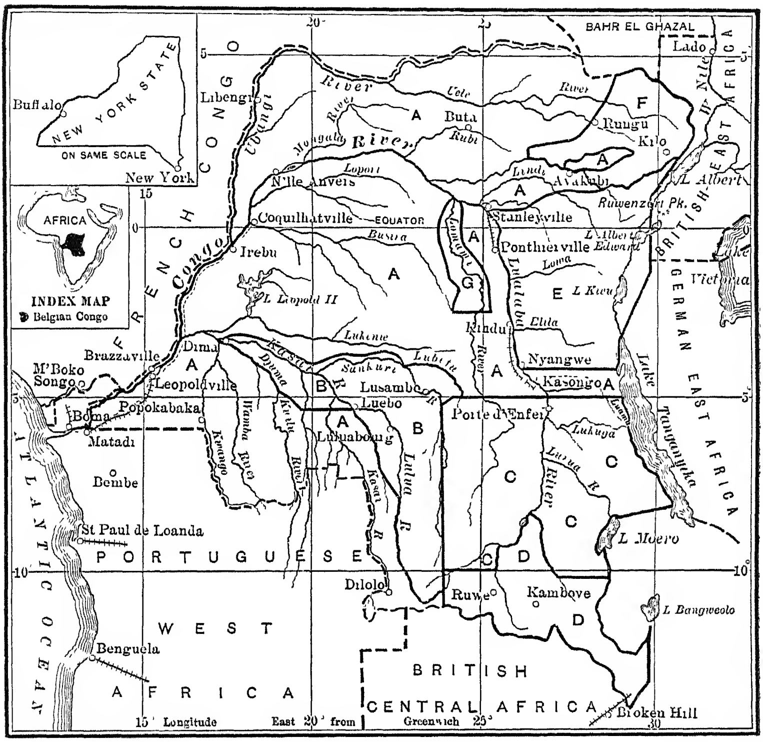
The actual production of copper or copper-ore merely awaits the arrival of the Cape-to-Cairo railroad, which has recently reached the Rhodesian frontier from the south, and presumably many of the deposits will be producing mines within less than two years.

b. The Bamanga Copper-Deposit.—This deposit, which was casually examined by one of us in 1908, lies on a small island in the Congo river, about 7 miles below Ponthierville. It was then and is now, according to report, being actively developed by a sub-company of the Chemin de fer des Grand Lacs.

The country-rocks seen consist of a red arkositic sandstone or quartzite, which is presumably younger than a coarsely-porphyritic biotite-granite gneiss with which it is in contact. With the granite-gneiss is a little hornblende-biotite gneiss, evidently representing a mashed basic igneous rock. These rocks dip steeply, the strike being roughly E–W. The ore occurs in parallel veins, cutting all the country-rocks, or what is equally probable, in short lenticular bodies, in fractures more or less continuous, and, in part at least, faults. These ore-bodies, as

exposed in the open-cuts, vary in width from 0.5 in. to 2 ft., and strike a little east of north.

The original minerals, filling the fracture-cavities and incasing the included fragments of country-rock, are quartz, chalcocite, and chalcopyrite. The first two are frequently well crys-



LEGEND

A—Société Forestière et Minière
B—Compagnie du Chemin de fer du Bas Congo au Katanga.
C—Comité spécial du Katanga.
D—Union Minière du Haut Katanga.
E—Compagnie des Chemins de fer des Grands-lacs.
F—Fondation de la Couronne.
G—Compagnie du Katanga.
Limits of Concessions.
Railroads
Telegraph or Telephone.

SCALE OF MILES
0 50 100 200 300 400 500

FIG. 2.—MAP OF THE BELGIAN CONGO, SHOWING RAILROADS AND MINING-CONCESSIONS.

tallized in vugs. The secondary minerals are azurite and malachite, with some native copper and bornite. Picked ore runs from 31 to 55 per cent. of copper, and appreciable quantities of gold and silver are reported to be associated with it. In November, 1908, the monthly product was said to be 3 or 4 tons, with an encouraging monthly increase. In the general

FIG. 3.—GOLD-MINING AT KILO.

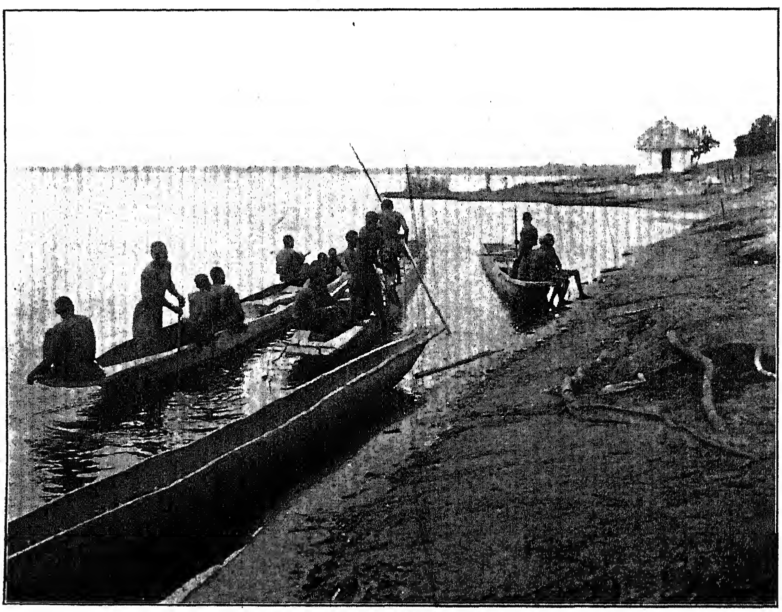


FIG. 8.—CANOE TRANSPORT, UPPER CONGO RIVER.

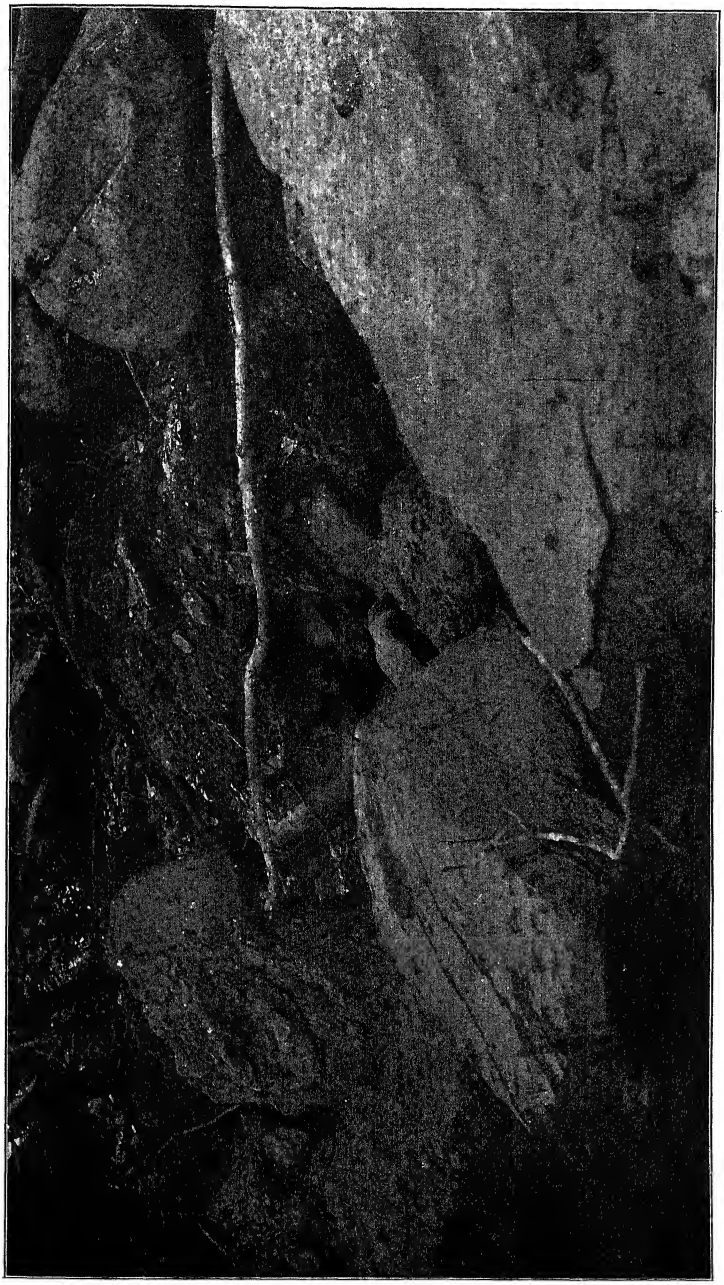


FIG. 4.—GNEISS BOULDERS COVERED BY RECENTLY-FORMED LIMONITE CONGLOMERATE, KASAI RIVER BANKS BELOW BANTUA SANKI. SUCH CONGLOMERATE IS THE CHIEF SOURCE OF THE IRON-ORE SMELTED BY THE NATIVES.

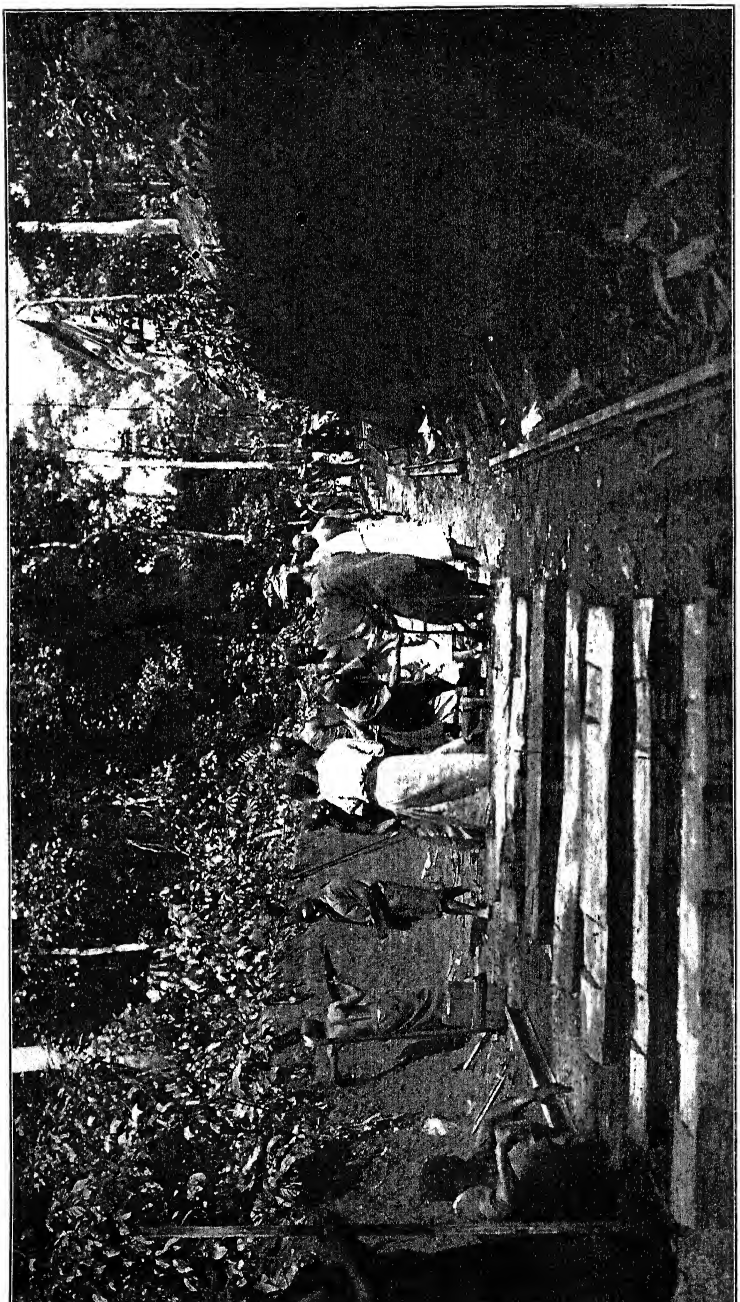


Fig. 7.—Railroad Construction, Grand Lakes Railroad.

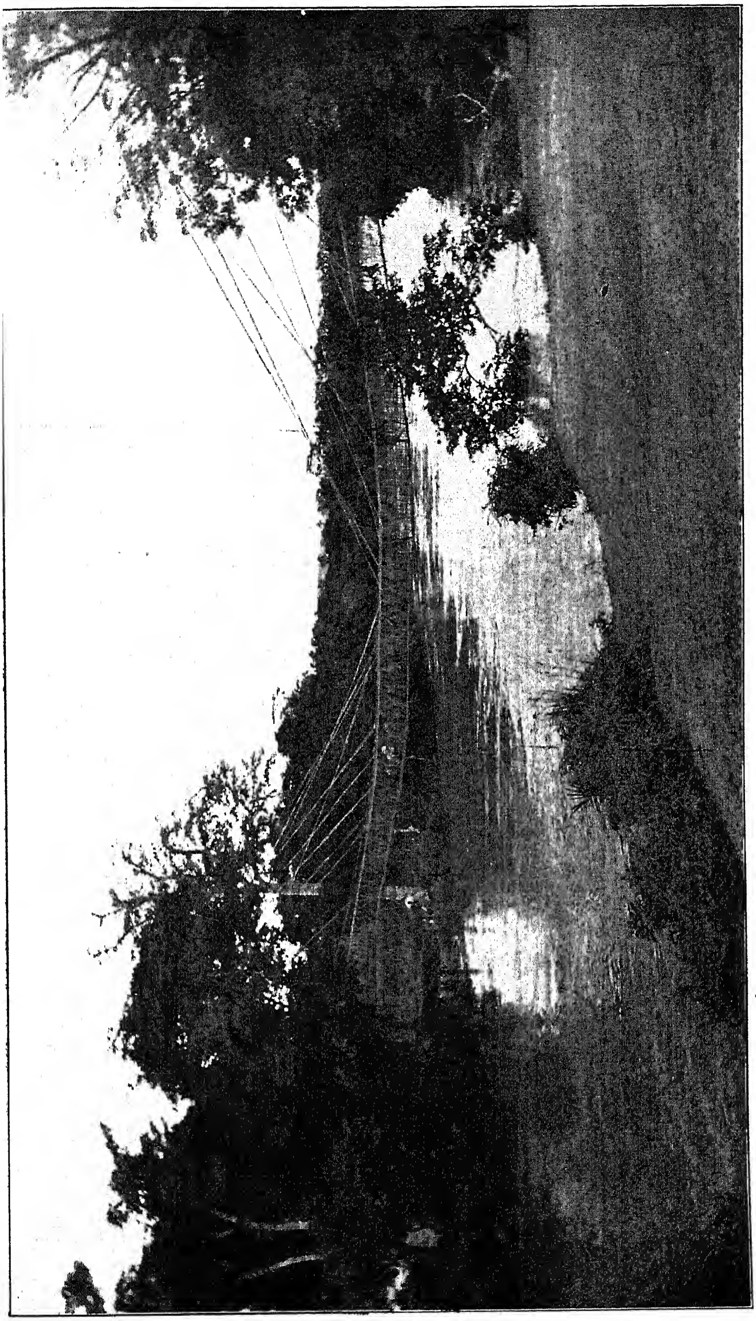


Fig. 9.—Vine Bridge Across Lubefu River, Lubefu.

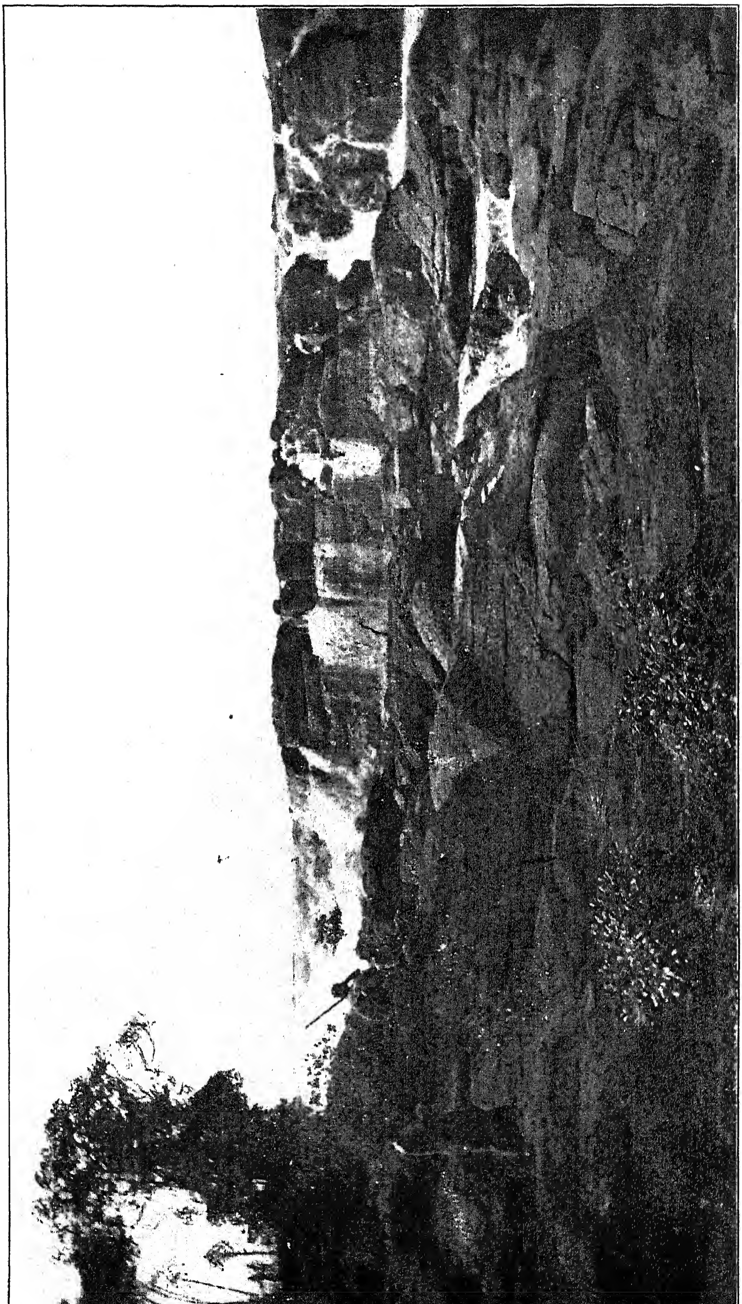


Fig. 10.—Eastern Part of Pogge Chute, Upper Kasai River, Mai Munene. Gneiss Outcrop.

vicinity, gold and silver occur in iron-stained fractures. In addition to small gold-placers near Bamanga, copper-placers are reported to occur along the river-bank, which was flooded at the time the mine was visited. Pebbles of rich copper-ore occur in these placers, in thin beds, inter-stratified with recent alluvial sands.

c. The Bembe Deposit.—Bembe[8] is situated in Portuguese West Africa (Angola), about 110 miles inland from the Atlantic coast and 75 miles south of the Belgian Congo frontier. The deposits, long worked by the natives, were the property of several neighboring villages, but natives from other villages were permitted to work them on a royalty. Untimbered holes, 3 or 4 ft. in diameter, were sunk to from 24 to 30 ft. depths, with native hoes and trade knives. Descent into and ascent out of these holes was by means of wooden pegs driven into the sides of the pits.

The country-rock consists of clay-slates and limestones, striking NW. and dipping steeply SW. The ore, largely malachite with some atacamite, occurs as masses imbedded in clays, apparently derived from the clay-slates. The lumps of ore, which has associated with it much iron oxide, vary in size from that of shot to masses weighing 3 tons.

d. M'Boka Songo, Mindouli, and Abikula Deposits.[9]—In the French Congo, just north of the Belgian Congo frontier, is a belt of copper-deposits, more or less continuous, stretching in an ENE. direction through a distance of 60 miles. M'Boka Songo and Mindouli, the better known of these deposits, occur respectively at the western and eastern limits of this belt. These two deposits have been mined by the natives for centuries, and even at the present day, in the dry season, from 300 to 350 natives are said to work at Mindouli. One pit for copper, made by natives at M'Boka Songo, is said to be 400 m.

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[8] Monteiro's *Angola and the River Congo*, vol. i., pp. 189 to 196 (1875). Thomas, Lewis, *The Geographical Journal*, London, p. 604 (1908).

[9] De Launay, *Les Richesses Minerales de l'Afrique*, p. 124 (1904). M. Barrat, *Annales des Mines*, Ninth Series, vol. vii., pp. 460 to 468 (1895). M. Levat, *Annales des Mines*, Tenth Series, vol. xi., pp. 1 to 65 (1907). M. Bertrand, *Revue Générale des Sciences*, pp. 792 to 796 (1895). A. Le Chatelier, *Comptes rendus des Séances de l'Académie des Sciences*, vol. cxvi., No. 17, pp. 894 to 896 (1893). Dupont, *Lettres sur le Congo* (1889). Brien, V., review of forthcoming publication on copper-mines in French Congo, *Tribune Congolaise* (Sept. 23, 1909).

long by 60 m. wide and 20 m. deep. Both copper- and lead-ores are smelted in small crucibles there. While malachite is the ore much preferred, chalcocite is recognized by the blacks to contain a high percentage of copper, and they add some of it to the charge. Shipments of malachite were formerly made to Marseilles, but heavy transport-charges rendered this unprofitable. At present, however, a narrow-gauge railway is being constructed between Brazzaville, on Stanley pool, and Mindoula, and active exploitation will soon begin.

There are two chief country-rocks in this hilly region: a limestone with a thin conformable sandstone layer above it, generally referred to as Devonian; and an unconformable reddish sandstone, accompanied by argillaceous mica-shales, fissile, and brick-red in color, supposed to be the equivalent of the Karoo sandstone of South Africa. Pebbles of the former country-rock have been found in the latter.

At M'Boka Songo the ores chiefly occur imbedded in clay pockets in the irregularly-weathered surface of the limestone. There are also more or less tabular deposits near the contact of the limestone and the conformably overlying sandstone. At Mindoula the tabular deposits at or near the contact predominate, and these ore-bodies coincide in dip, as a rule, with the country-rocks. The limestone, in the vicinity of the ore-bodies, is silicified, and in instances is a veritable stockwork of veinlets of chalcocite. Sheets of ore follow down the joints in the limestone.

The ores present are numerous and many of them unusual. They include malachite, azurite, dioptase, chalcocite, tetrahedrite, and black copper oxide, argentiferous galena, cerussite, and wulfenite, calamine and willemite, limonite and an iron phosphate, and native silver. As gangue minerals, calcite, quartz, and a carbonate of manganese are present.

At M'Boka Songo the country-rock is highly calcareous, while at Mindouli and Abikula it is more siliceous. As a consequence, through mass chemical action, silicates predominate at Mindouli and Abikula and carbonates are the main ores at M'Boka Songo.

As is true of many mineral deposits where the workings extend to no great depth, the origin of these ore-bodies is a mooted question. Barrat and most of the older investigators

believed them to be the oxidized portion of veins, the secondary minerals replacing the limestone to some extent. He believed the deposits to be genetically related to the intrusion of a diabase which cuts the limestone in this general vicinity. Levat believes, as does apparently Brien, that the minerals were originally deposited in the overlying conformable sandstones, and that, through leaching, concentration, and re-deposition, they are now found in the limestone, the composition of which is favorable to replacement.

3. *Tin.*

While cassiterite has been found at several and reported from many localities in the Belgian Congo, it is only in the Katanga[10] that deposits rich enough to exploit in the future have been found. The tin-belt of the Katanga is situated on the west slope of the rugged Bia mountains, and extends NE. from a short distance north of Ruwe to a point not far south of Lake Kabele through a distance of about 110 miles. The rock of these mountains is a biotite granite, which to the west intrudes mica-schists, slates, and quartzites, carrying tourmaline. The sedimentary rocks are considered by Studt[11] to be of Silurian age.

The tin-deposits are of three kinds: 1, quartz-muscovite-cassiterite veins; 2, residual deposits from these; and 3, placers derived directly from the other two sorts of deposits. The veins for the most part are situated in the sedimentary rocks near the granite, but sometimes are in the granite itself. They are vertical and occur in two sets, one quite regular in values striking NW., the other at right angles, sometimes richer but less constant than it in content. The veins are from 300 to 4,000 ft. long. Cassiterite occurs in good and often very large crystals, imbedded in quartz, and is frequently more abundant near the borders of the veins. Undoubtedly these veins are dependent in origin on the after-effects of the granite intrusion.

The residual deposits, most important economically, are derived from the breakdown of these veins. The tin-bearing

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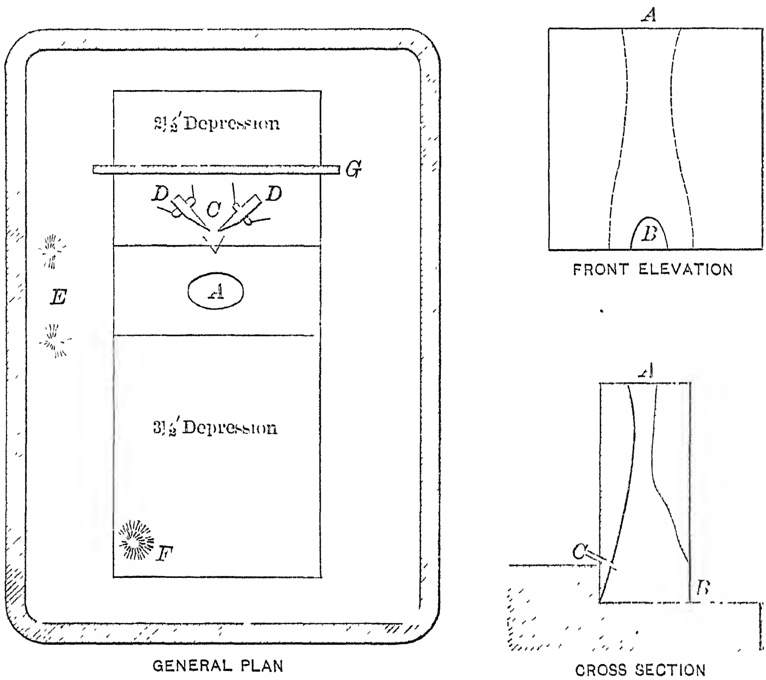
[10] Buttgenbach, *Annales de la Société Géologique de Belgique*, vol. xxxiii., Mémoires, pp. 49 to 52 (1905-06). Buttgenbach, Carte Géologique du Katanga, *Annales du Musée du Congo*, Second Series, Katanga, pp. 70 to 72 (1908). Various reports of the Tanganyika Concessions, Ltd. (1900 to 1906).

[11] Studt, *Annales du Musée du Congo*, Second Series, vol. i., p. 14 (1908).

layer averages 2 ft. in thickness, and carries a mean tenor of 1 per cent. of tin. These deposits, discovered in 1903, are estimated to contain 20,000 tons of tin of excellent quality. On the ground, 8 tons of tin were smelted early in 1906.



A. Feed-door.
B. Discharge-door.
C. Tuyere.
D. Bellows.
E. Ore and charcoal.
F. Clay.
G. Pole for bellows-men.

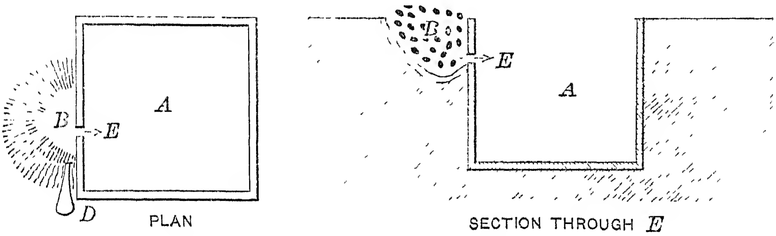
Fig. 5.—Nondo Primitive Furnace.

The deposits are close to a navigable stretch of the Lualaba (Congo), and plenty of water-power is at hand.

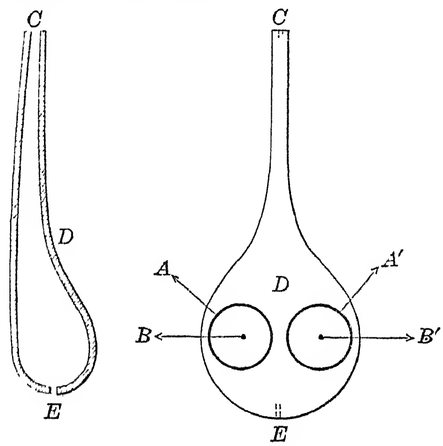
4. *Iron.*

Iron-ore of various grades is plentiful in the Belgian Congo, and indeed in that country is one of the world's more important iron-reserves. Since long before the white man's coming, Congo natives of many tribes have smelted iron-ore and skillfully fashioned the metal. The craft is highly rated among the natives, and not rarely descends from father to son, or at

least remains forever within a certain caste. The ore is derived largely from a recent iron-stone conglomerate, Fig. 4, which is widely distributed in the Belgian Congo. The limonite and hematite cement of this conglomerate is readily smelted by the natives, but is of too low grade and occurs in bodies too small for exploitation by the white man.



A. 3- by 3- by 2.5-ft. clay-lined *buh*, or slag-collector.
B. Shallow pit; open-hearth furnace in which are placed ore and charcoal; clay lined.
C. Tuyere; clay cylinder.
D. Bellows.
E. Opening through which slag escapes.



Sketch of bellows. *A* and *A′*, loose leather caps, to be worked up and down by handles *B* and *B′*, compressing air in wooden shell *D*, which is expelled at orifice *C*. Valves at *C* and *E*.

Fig. 6.—Djuma Kwilu Iron-Furnace.

In nearly every part of the Belgian Congo are primitive iron-smelters, and in principle they are very similar to either the shaft or open-hearth iron-smelters in civilized countries.. The accompanying sketches, Figs. 5 and 6, show in a general way the characteristics of these furnaces, which are types used

respectively by the natives of the Djuma Kwilu region and of the mountains NE. of Kasongo.

The furnaces are in each case charged with small fragments of limonite and hematite, mixed with wood-charcoal, or in alternating layers with it. The blast is furnished by hand by means of a bellows constructed in general after the principle outlined in Fig. 6. The furnaces are usually clay-lined or are constructed directly in a huge ant-hill. The tuyere is of baked clay. The product is a more or less porous iron containing considerable slag and some charcoal. This iron is placed on a stone anvil, where it is freed from its impurities by pounding, and is then cleverly forged into knives, spears, bracelets, hoes, etc. Apparently the whole process depends for its success largely upon the benign oversight of the iron-maker's fetish, which is ever present when iron is to be made.

Among the Nondos, living to the east of the Lualaba (Congo) river and about 50 miles NE. of Kasongo, iron-smelting is highly developed. The natives, with pointed sticks and iron spear-heads actually mine an impure hematite in shallow "gopher" holes and inclines. A part of the ore, a specular hematite, is obtained at Mauna Kusu, 30 miles away, and towards Kasongo. The chief smelters are located at Nyanga and Lumbukahula.

In the forests nearby, beneath rude thatches, logs, half-buried in the soil and partly sand-covered, mark the locations of charcoal "ovens." Long before reaching one of these villages the "puff-puff" of the bellows is heard, and upon arriving the well-built houses and comfortably-clothed natives speak of a prosperity unusual in Central Africa. In each village are several smelters, each inclosed in a clay house with banana-leaf-thatched roof. These buildings are 15 by 25 ft. in plan, with 8-ft. walls and a ridge 7 ft. higher.

The burnt-clay-lined furnace, as seen in Fig. 5, is roughly a blast-furnace of the shaft type on a small scale. The charge is, in part, made through the front opening, but largely through the top of the furnace. The front opening is closed during smelting by a large flat stone, clay-sealed, with the exception of two holes on either side for the escape of molten slag and iron. The charge, of 0.25-in. fragments of iron-ore and charcoal, is placed in alternate layers in the proportion, by bulk, of

one to four. Two runs are made per day, one from 5 to 11 a.m., the other between 1 and 7 p.m. Each run results in about 200 lb. of porous iron, which is hammered into a solid mass free from slag and charcoal. This in turn is forged by the smiths into hoes, knives, etc., which are much prized throughout the country.

Several natives tend the furnace, besides the two sweating, smoke-and-dirt begrimed bellows-men, whose job is not an easy one. All strenuously affirm that the Arabs had no influence upon the construction of their furnaces, but claim their knowledge of smelting and iron-working was a gift of the gods. It seems highly probable, then, that this trade was practiced among the Nondos before the Arab invasion. Before the white man's arrival, smelting was preceded by sacrifices of goats and chickens, to propitiate the gods and to assure a large yield. Whispers are also heard that even human sacrifices were not unusual.

In the Djuma Kwilu region, at and near Kivula, the natives use the type of furnace sketched in Fig. 6. Excepting in detail, the smelting- and working-processes are the same as with the Nondos. The latter work on a considerably larger scale, however, and near Kivula the type of furnace is distinctly "open-hearth."

There are, in the older rocks which rim the Congo basin, many deposits of hematite and magnetite of excellent grade, some of which are, in magnitude, to be compared only with the iron-deposits of the Lake Superior region. Our knowledge of these is at present, however, too meager to enumerate all of the classes of iron-ore deposits represented. Some are evidently magnetites of igneous origin (and related to gabbroic magmas); others metamorphosed sedimentary rocks, resembling the hematite-magnetite deposits of the Vermillion and Marquette ranges; again, others are replacements formed during the dying stages of granitic intrusions, and still others probably originated through the recrystallization and migration of material during regional metamorphism.

That huge iron-deposits exist in Central Africa is certain, and, in our opinion, before the end of the present century this region will become an important factor in the world's iron-production.

5. *Salt.*

In eastern Belgian Congo are a number of hot and cold saline springs, from which the natives have long obtained salt, an article highly prized among them. At Bena Sombo, near Nyangwe, rather strongly saline water, ranging in temperature between 44° and 47° C., bubbles gently from crevices in flat-lying shale at the wide head of a valley. The process of manufacture, which has been followed by the natives for generations, and has even made an important personage throughout the country of the chief, is as follows: Once a month, having insured an abundant flow of unusually saline water by the sacrifice of a goat to the fetish of the place, the villagers build around the springs low mud-and-rock embankments, thus impounding the water. From these ponds round earthenware pots are filled and carried to the village, where the salt weed, "medicine" to hasten evaporation, is thrown into them and they are placed over fires. The resultant salt is medium-grained and dark gray, due to the contained smoke and other impurities. Some 250 lb. is produced monthly, and, strange to say, this salt is generally preferred by the natives to the pure salt imported from Europe.

Years ago salt was obtained from four or five nearly-boiling and more or less intermittent hot springs, issuing at the extreme north end of Lubware peninsula in Lake Tanganyika. Due to some superstition, this work was abandoned, and it is now difficult to get the natives of the peninsula even to go near the springs. A liter of this water yields a heaping teaspoonful of fairly-pure salt.

Where such salt-springs do not occur, the natives in swampy places cultivate, or gather wild, certain grasses and water-plants which contain considerable potassium chloride. By burning these plants and permitting water to seep through the ashes, and then evaporating this water, potassium chloride is obtained, which serves as a substitute for salt.

6. *Silver and Other Metallic Products.*

Native silver, while present in association with the copper-ores of Mindouli, has not as yet been found in quantity in the Belgian Congo. The rarity of Tertiary and post-Tertiary igneous intrusions, with which silver-ores are so often associated in other countries, may explain this. With the exception of the occur-

rence of platinum and palladium in association with gold at Ruwe, these metals have not yet been definitely reported from the Belgian Congo. Lead- and zinc-ores occur sparingly with the copper-ores in the French Congo, and lead was there smelted by the natives.

7. *Non-Metallic Products.*

Up to date neither petroleum nor merchantable coal has been found in the Belgian Congo. Of the other non-metallic minerals, diamonds, salt, barite, allanite, copal, marble, and building-stones occur in more or less quantity. While diamonds have been found in the Katanga and elsewhere, there is no proof as yet that this precious stone occurs in workable quantity. Copal, in part fossil copal, and in part resin from present-day trees, is bought in small quantities from the natives by several commercial companies. In 1908 its export-value from the Belgian Congo was $358,000. The other non-metallic minerals are not exploited.

VII. General Mining-Conditions.

Transportation.

Matadi and Boma (from 18 to 20 days from Antwerp and Bordeaux) are at present the principal ports of entry for the Belgian Congo. Steamers leave for these ports from Antwerp every three weeks (Belgian Line) and once a month (Holland Line); from Bordeaux once a month; from Liverpool for the African West Coast every week; and from Lisbon to Cabinda, etc., every two weeks.

The northeastern corner of the colony is, however, served in part by the Nile steamers and railroad (Alexandria to Lado, 35 days), while the eastern frontier receives some supplies and mail by the Indian ocean ports, Dar-es-Salaam (German East Africa) and Mombasa (British East Africa). The trip by rail, steamer, and caravan, across British East Africa, requires from 30 to 35 days. Now that the Cape-to-Cairo railroad extends from Cape Town to the Belgian Congo frontier, it will naturally become more and more the outlet for the rich Katanga country. The railroad from Benguela bay (Portuguese West Africa), when completed into the Katanga country, may be a strong competitor of the Cape-to-Cairo road.

Within the Belgian Congo itself there are now three rail-

roads. The Mayumbe railroad extends 30 miles N. from Boma, and the lower Congo railroad connects Matadi, the seaport, with Leopoldville at the beginning of the Upper Congo navigation. This narrow-gauge road, which is 250 miles long, handles at present practically all of the exports and imports of the Belgian Congo. The Congo river is navigable from the terminus of this railroad (Leopoldville) to Stanleyville, a distance of 20 days by steamer, at which point the first division of the broad-gauge Grand Lakes railroad begins, Fig. 7, running to Ponthierville, about 80 miles up river. From the latter point there is a stretch of navigable river to Kindu, four days by steamer, where the second division of the railroad starts, which is now completed to a point opposite Kasongo. This division will be completed to Porte d'Enfer, above which the Congo is or can be made navigable to Lake Upemba, where a third division will begin, which, when finished, will complete the connection between the port of Matadi and the Katanga region.

Another railroad-line is being surveyed from Pania Mutambo—just above Lusambo—to the Katanga mining-region. In addition to these railways, a traction-engine route is under construction in northeastern Belgian Congo, which will connect the navigable waters of the Congo with those of the Nile.

Many steamers belonging to the government, to commercial companies, or to missions, ply the navigable stretches of the Congo just described. Two of these steamers, between Leopoldville and Stanleyville, are of 500 tons. In addition to the 2,000 miles of the Congo which are navigable, there are about 5,000 miles of navigable tributaries, among which are the Kasai, the Sankuru, the Ubangi, and the Lomami. One may ascend many smaller tributaries of these affluents of the Congo in canoes, Fig. 8, or whale-boats.

Interior travel is, however, very largely on foot by caravan-trails, which are broad and usually kept in good condition. Fig. 9 is a view of a suspension-bridge, constructed of vines by the natives, on a caravan-route crossing the Lubefu river, which at this place is 300 ft. wide. Off of these main routes the native trails are often difficult to travel, being narrow, swampy, overgrown with grass, or almost impassable through forest. It is over these trails that one's baggage, supplies, and provisions

are carried by native porters. When they are heavily charged and are expected to carry steadily for a week or more, 12 miles is a good average day's journey, though at times double that distance is made.

The postal service in the Belgian Congo is, all things considered, very satisfactory. There are two main telegraph- and telephone-lines within the colony; one connecting Boma and Coquilhatville, the other Kasongo and Uvira. Cables to Europe usually go via Brazzaville and Loanga in the French Congo.

2. *Labor.*

There is an estimated population in the Belgian Congo of from 12,000,000 to 15,000,000 natives. It seems, therefore, at first sight, that the labor-problem is an easy one, as certainly the major part of the manual labor must be performed by natives. Such, however, is not the case. The black is, as a rule, lazy and perfectly content to allow his wife, or wives, to work in the plantations to insure him the life of idleness that to him is one of luxury and happiness. The dignity and the value of labor are, nevertheless, being slowly taught the native, and as he has by nature a strongly-developed imitative faculty, the labor-problem is by no means hopeless. Clever native carpenters and blacksmiths are seen at many of the State posts, and unusually well-trained Congo men have made excellent clerks and good locomotive-engineers. If properly handled, native labor is not too difficult to obtain in many parts of the colony, and under good head-men, the Congolese is a fairly-efficient laborer. For skilled labor and comparatively unimportant clerical work, the Sierra Leone, and particularly the Senegalese, negroes are very satisfactory, as are also the East Coast black men.

The climate greatly handicaps the work of the white man in most parts of the Congo, and in manual labor he can never do more than direct the work of natives. Even with the best of luck in any kind of work, mental or physical, the white man's efficiency, counting illness and the naturally enervating effect of the climate, is cut down to about one-half of what it is in a temperate climate.

The usual term of service of government employees is 35 months in the colony and 6 months in Europe, though many of

the companies require of their agents but two years in the tropics. Apparently, a man between 25 and 35 years of age stands the life the best. This simply means that a man must be in the best condition, physically and morally, in order to do good work in the African tropics with the least permanent harm to himself.

3. *Timber and Water-Power.*

At the present time, wood is the universal fuel in the Belgian Congo, excepting for driving the locomotives of the Lower Congo railroad. Nearly everywhere, except where the savanna is most characteristically developed, wood for fuel is abundant. For important mining-operations, however, hydro-electrical installations will undoubtedly be used. Wherever the older folded rocks occur, water-falls and rapids are common, and power can be developed in quantity very easily. Fig. 9 is a photographic view of the eastern part of the Pogge chute, on the upper Kasai river. The falls have a head of about 30 ft., and about one-third of the total width is shown in the view. Of all the continents, Africa will benefit most by the perfection of electrical smelting.

4. *Costs.*

It is impossible to give general costs for mining-operations in the Belgian Congo, and the costs in particular cases will vary so radically under changing conditions that any figures that could be given must be of little value. It may, however, be said that costs are high and that many mineral deposits in the Belgian Congo, at present valueless, would be paying mines in practically any other country.

Transport-costs are excessive. To get machinery into the upper Congo region or the lake country from Antwerp, one must count on at least twice the original cost for transport and customs duty.

A constantly-changing white personnel is an expensive item; for outside of the salary, which is naturally high, each up-country agent is paid at least six months out of his term when he is *en route* to and from Europe. The journey itself is expensive, amounting to about $1,000 for each up-country agent, and his actual living-expenses in the colony amount to about $1,200 per year.

Native labor is comparatively cheap, but it is also comparatively inefficient. Two ordinary negroes will sink a pit 2 ft. in diameter in fairly incoherent medium-sized gravels and clay or sand to a depth of from 4 to 6 ft. in a day of 10 hr. A man's pay ranges from 4 to 7 cents per day, plus his food, which costs about 3 cents per day. The wage of a Sierra Leone or Senegal negro ranges from 50 cents to $1.50 per day, including his food, though the average pay is about 65 cents.

For fresh food the white man must depend largely on goats, sheep, and chickens, which can generally be readily purchased from the natives, along with a few fresh vegetables and fruits. In the dry season game is usually plentiful, and the State posts generally have excellent gardens, from which European vegetables are obtainable. Unfortunately, however, the greater part of the white man's food must be sent out from Europe.

As to the safety of capital invested in the Belgian Congo, that country is much safer than the average South American republic. Within limited areas revolts may take place, but neighboring tribes cannot set aside their own differences long enough to unite against the white man. Moreover, even when a territory is in revolt, in many instances, the white man's property is held sacred. Evidence of the unsavory stories of Congo atrocities published from time to time in American and English newspapers was not seen by us in our two years' journey throughout the Belgian Congo.

The Conditions of Accumulation of Petroleum in the Earth.*

BY DAVID T. DAY, WASHINGTON, D.C.

(Pittsburg Meeting, March, 1910.)

IN 1897 I published a proposed explanation[1] for the variation in color and specific gravity of Pennsylvania oils. A *résumé* of this subject was also presented at the First International Petroleum Congress, in Paris, in 1900. Since that time

* Published by permission of the Director of the United States Geological Survey.

[1] *Proceedings of the American Philosophical Society of Philadelphia*, vol. xxxvi., No. 154, pp. 112 to 115 (January, 1897).

much experimental work bearing upon this explanation has been carried on by me, in the United States Geological Survey, and under my direction, by Dr. J. Elliott Gilpin, Marshall P. Cram, and O. E. Bransky, at the Johns Hopkins University. The experimental results thus accumulated seem to throw considerable additional light upon the variations in color and specific gravity of petroleum, as found in different parts of the world, and also upon the variations in chemical composition. Therefore, it seems as if a summary of the evidence thus obtained might be of value to those members of the Institute who are interested in the examination of oil-fields. The summary is here presented for that purpose.

The phenomenon has often been observed in many oil-fields that oils of different color, specific gravity, and chemical composition may be found within short distances of each other. A conspicuous example of this is the occurrence at Jennings, Welsh, and Anse la Butte, La., of petroleum chemically somewhat similar to Russian petroleum; that is, characterized by the presence of naphthenes. Another characteristic is the presence of considerable percentages of asphaltic compounds, sufficient in quantity to disguise the presence of solid paraffin wax, and perhaps also to render that paraffin wax more soluble in the oil than it would be were not these asphaltic compounds present. At Belle Isle, La., only 40 miles from Anse la Butte, petroleum is found unusually light in color, about that of sauterne wine. This petroleum is practically free from sulphur, very much lighter in specific gravity than the other oils, free from asphalt, and shows abundantly the presence of paraffin wax in solution. At first glance the oils seem to be entirely different, but chemical examination shows them to be as closely related to Russian oils as are those from Texas, and markedly different from the oils of the paraffin-hydrocarbon series found in Pennsylvania.

Other instances are noted of similar variations in oils within short distances, especially in Mexico, as well as in California, Wyoming, Oklahoma, Texas, and elsewhere. The oils usually, however, contain some characteristics in common, which would indicate that one oil has been derived from the other. In endeavoring to express such a genetic relationship, it has been supposed that the dark oils have been derived from the lighter

ones by evaporation and oxidation. In the instance cited for Louisiana, this is evidently not the case, since the dark oil contains much material not found in the light oil, and which never could have been obtained from the light oil by oxidation. I refer to the considerable amount of sulphur present in the dark oil and not in the light oil. If we add to the light-colored oil sulphur compounds and asphalt in appropriate amount, the dark oil is practically reproduced. That the dark oil is not an evaporation-product from the light oil is shown by the fact that it contains hydrocarbons as volatile as those of the light-colored oils, though not in so great a quantity. The proportions of the hydrocarbons present do not admit of working out any probable process of evaporation which would take out some of the heavier constituents of the light oil and leave the percentages of very volatile ingredients which are found. There is, however, no evidence that the light oil may not have been obtained by some process from the darker oil, and long ago the explanation was offered that the lighter-colored oil was produced by some process of distillation from the darker oil. This explanation is practically impossible, inasmuch as the light oil contains heavy paraffin wax, and oils with high boiling-points, while the dark oil also contains volatile oils, which would have distilled off long before some of the oils found in the light-colored material.

If there is a relation between these two oils, some change has taken place by which oils in solution in the dark, more complex oil, have been left behind, while other ingredients of the dark oil have been separated to form the light-colored product.

In my paper I referred to the observation made in a vaseline factory, that when a black oil containing some pitchy material, very black in color and semi-solid in consistency, is heated so as to be perfectly fluid and is then allowed to diffuse slowly through dry fuller's earth, the first diffusion-product is lighter in specific gravity than the original material. It is perfectly colorless, and remains entirely liquid at the ordinary temperature. On experimenting with this material, it was evident that the lighter oils contained in the original materials diffused more rapidly through the fuller's earth—just as hydrogen will diffuse through a porous medium more easily than oxygen will, and therefore separates itself from oxygen. The idea occurred that

crude oils in traversing dry, fine-grained shales in the earth might also diffuse with unequal rapidity, resulting in differential diffusion of the products; that is, fractionation by diffusion. Experiments clearly showed this to be the case with specimens of dried Devonian shale from Pennsylvania.

Clifford Richardson[2] applied this experiment to Texas oil, and found that sulphur compounds in that oil are left entirely behind by failing to diffuse into fuller's earth.

A long series of experiments undertaken in the U. S. Geological Survey showed that when a glass tube is packed tightly with dry fuller's earth, and one end is allowed to stand for one or two days in crude petroleum, the oil diffusing up through the clay fractionates to a considerable extent. Thus, when the upper fifth portion of the clay is dropped into water, a lighter-gravity oil is driven out by the water. This first fraction is entirely colorless. The lower fractions are heavier and more and more highly colored. The bottom section may be almost solid, and is darker in color than the original oil. The upper portion, although lighter, yields on fractionation some heavier oils and even paraffin wax, but the proportion of lighter oils is greater than in the original and much greater than in the lowest sample. A fractionation by diffusion has been effected, which is similar to fractionation by distillation, but is not so complete.

It has since been shown by Dr. C. Engler[3] that no chemical change is effected in this process, but merely a mechanical separation of liquids from liquids.

It should be noted that when the clay saturated with oil is stirred with water in order to drive out the oil by water, another partial fractionation is effected. The heaviest oil remains in the clay with the water to the extent of a third or a fourth of the entire oil, while the rest leaves the clay.

Another interesting feature of this diffusion is that the oil will not only rise to a height of 5 or 6 ft., for example, in a tube packed with fuller's earth, directly against gravity, just as it does in a wick, but if the tube be sealed at the upper end the oil will still rise in the tube, driving out the air in the pores of

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[2] *Journal of the Society of Chemical Industry*, vol. xxi., No. 5, pp. 316 to 317 (Mar. 15, 1902).

[3] *Zeitschrift für angewandte Chemie*, vol. xiv., No. 36, p. 889 (Sept. 3, 1901).

the earth and compressing it in the upper portion of the tube with a pressure sufficient to blow out the clay if the top of the tube be suddenly broken off and the air thus released. This effect of capillarity seems to produce pressures similar to those obtained in measuring osmotic pressure—which have been carried to about 40 atmospheres. The most common pressures found in oil- or gas-wells in Pennsylvania and West Virginia agree strikingly well, although occasional natural-gas wells of much higher pressure have been reported.

As stated above, water drives the greater part of the oil out of the clay, though not all. Conversely, oil will not enter wet or even moist clay. In fact, it has already been pointed out by me as well as by L. Mrazec, Director of the Geological Survey of Roumania, that wet, porous rocks are the most impervious cap-rocks for oil, so long as they do not have cleavage-planes in which the space is greater than capillary.

The size of the capillaries is of great consequence in these diffusion phenomena. Finely-divided material, such as amorphous silica, has no observable fractionating-power on oils that are readily fractionated by dry clays. Naturally, ordinary sands neither fractionate nor do they allow water to drive out oil from them with the same pressure phenomena as are shown in clays.

From these phenomena it is easy to derive a probable method of accumulation of oils in the sandstones of Pennsylvania, if we only assume that dark-colored oil (possibly containing sulphur and asphalt) entered shales varying in fractionating-power, due to varying porosity or moisture. By this means the oil would remain absorbed in the shales without forming an available pool until the entrance of moisture from any direction had the effect of driving the oil to such places where the expelling action of water is the slightest; that is, in the sandstone layers between the layers of shale. Considering the vastly greater power of capillary action over gravity, it makes no difference whatever whether the water drives the oil up or down. It will expel the oil with greater or less pressure upon the contained natural gas, until there is formed a pool of oil of light or dark color and under greater or less pressure compared with adjoining pools, where the conditions vary with the character of the shales, which vary in moisture and porosity from one acre to another.

The age of the shale has nothing to do with the accumulation process, provided that the shale has had time enough to lose sufficient original water to admit the oil.

Obviously, the locus of the oil pool will be a layer of sandstone, because water has least power for driving it out where the capillaries are large. If, on the other side of the sandstone, the shale is dry, the oil is absorbed and dispersed until it reaches the surface or is hemmed in by wet shale which it cannot enter, for the same reason that it cannot re-enter the shale from which it has just been driven by water. There the oil must accumulate. How long can it remain there? Indefinitely? I think not; for the driest shale was once mud, and if by slow process the shale does not become too dry to hem in the oil, it has all the chances in favor of the formation of minute cracks, which are not oil-tight. These cracks are not oil-tight unless absolutely sealed with water.

It seems very improbable that oil could have existed since Devonian time in the present pools.

These diffusion phenomena make much more probable the proposition advanced many years ago by Dr. John N. MacGonigle, of Oil City, that the oils are the same as the Ordovician limestone oils of Ohio, with the sulphur removed by diffusion.

One further observation. It was found in my early work, which has since been verified by a long series of quantitative experiments by Gilpin and Cram at Johns Hopkins University,[4] and again by Herr,[5] in Russia, that unsaturated hydrocarbons are less diffusible than saturated hydrocarbons even when less viscous than the saturated ones, and hence they are left behind. This seems to explain very satisfactorily the fact that these diffused oils of Pennsylvania are practically entirely saturated oils, in strong contrast to the undiffused oils of California, and many other pools in the United States.

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[4] *Bulletin* No. 365, *U. S. Geological Survey*, p. 26 (1908).

[5] *Petroleum*, p. 1284 (August, 1909).

Systematic Exploitation in the Pittsburg Coal-Seam.

BY F. Z. SCHELLENBERG, PITTSBURG, PA.

(Pittsburg Meeting, March, 1910.)

SYSTEMATIC exploitation in the Pittsburg coal-seam on a large scale is simple where the boundaries of the property do not interfere by forcing drainage-, ventilation-, and transport-lines of entries to be run to particular confined fronts. A prime condition governing development is the natural direction of the seam's perpendicular cleavage-planes, giving long lines of smooth faces near together and parallel, bearing about 65° to the left of the meridian and therefore crossing the basins. The gradients of dip into the basins change from 10 per cent. at the east rim to practical flatness northwestward across the great coal-fields, a hundred miles wide; the dip being halved in rate at each successive crossing-over of an anticlinal crest. Elevations above sea-level on this transverse section, in scope of present development, differ by 1,000 ft. between low trough and high outcrop. Local irregularities of dip are found, but the main feature to be recognized is the pitch of the basins—the general rise northeastward at 20 ft. per mile, from 100 ft. below sea-level at the low point near the SW. corner of Pennsylvania to 1,500 ft. and more above tide, and the appearance then, at the surface, of the lower coal-seams carried in the next deeper-lying thousand feet of rock-measures; below which series come the oil- and gas-sands of Oliphant's charted column of strata, reaching down altogether 3,000 ft. below the Pittsburg coal-seam.

Because of the general regularity of occurrence of the coal-seam, the mine-map depicting the courses of economic operation to be undertaken and the steps in controlled exhaustion, need be but a projection of two dimensions on the horizontal plane of space, and this for the most part composed of natural, almost unbroken lines, on the face and on the end of the coal, co-ordinating rectangularly.

The thin-vein gas-coal of the Pittsburg region is bright, cuboidal, and hard, and is marked by having the mineral matter mostly in bedding-slates defining the measures, while the coking soft coal of tall cross-section at Connellsville is columnar-fissured to incoherence, and has its mineral matter less segregated. Intermediate in characteristic appearance and position is the thick coal of the transition territory.

The bedding layers, the main measures of the seam, which provide the greater part of the product, are very uniform in a district, the extremes of gradual variation in size being from 9 ft. total thickness at the east near Connellsville to barely 4 ft. 6 in. at the west beyond Pittsburg. In the eastern basin, 5 in. of main coal at the top and 2 in. at the bottom are left standing to cover the fire-clays above and below the bed and protect them from air-slacking. In the western region several inches of slaty bottom coal are left standing; then the 1-ft. fire-clay "draw-slate" is regularly taken down as a hanging clod, in order to secure a safe, smooth, and higher top under the roof coal, which is of slaty layers and not to be removed unless ripped in grading entry over "swamp" in the seam's lay. Normal to the cleavage-planes, on the face of the coal, is the direction of attack easiest to get product in quantity, and so in flat districts the rooms are always run for depths of 75 yd. The last detail of survey is the setting in the roof of "sights." These are points for ranging the room-driving on parallel lines, with the dividing pillars, the room ribs, of even thickness.

The technical "mining" requisite for proper preparatory work at the face of advancing workings is to cut loose, by channel-trenching, by hand or power pick, the bottom and one side at least, before rending violently the body of the coal to the same depth from the face. Formerly, in the gas-coal, its peculiar band-slates were removed in making along them, instead of at the bottom, the undercutting channel called the "bearing-in"; and the vertical kerfing at the side called the "shearing" also always was made for getting lump coal by wedge and sledge instead of the "all-digging" that answered for soft coal. It is now attempted partly or completely to supersede trenching and shearing by heavy-charge blasting; but experience in every locality will repeatedly call for better select "lump," hard-coal product, with less dangerous shooting.

The method of mining by all-pick work is used now only at smaller mines among the outcrop-areas of the seam. By it the miner's unit of work for 12 months was, singly, to drive up a 7-yd. wide room and then bring back from the head a continuous 4-yd. rib. He had to set three rows of posts and lay wooden track going up, and could bring out most of the wood after the double use of some posts and of extra rib-posts.

The diameter of the ordinary post is 1/20 of the height; and the cap-piece, 18 in. long, 2 in. thick, and 4 in. wide, wedges the post over the top.

The posts are placed in 5-ft. spans, to hold the roof-plies from raveling. Heavier timber is usually needed only where the entry opens over more than the common scant 3-yd. width required between continuous trim coal pillars to keep a track and its drain in proper order.

The coal-mines of the Pittsburg seam are symmetrical in layout of entries, aside from gravitating drainage and transport main-lines. In the gas-coal districts, on light dips, the cross-entries are a pair of butts in a belt 100 ft. wide, located at intervals of 500 or 600 ft. along the section-entries, which are in triple, comprising a pair of return air-course entries, with main haulway between, on the face, in a belt 200 ft. wide, disposed about every 1,400 ft. along the trunk-belt. The trunk-belt is made 400 ft. or more wide, so as to contain the main entries for all purposes.

In the panels, which are 400 or 500 ft. on face by 1,200 ft. along face, the room and rib working is from the opposite butt-entries, but not always simultaneously for rooms to meet midway, as it may prove better to bring the roof down to settlement after the immediate retreat-work of drawing out the ribs, in the 200 ft. half-space, say NE., before starting in from other tier SW., there being then shorter roof-break lines, diagonal half end and half face along the rib-stumps, and also more solid coal next as a barrier against settling-falls. Usually, however, the rooms turn off both sides at once from the pairs of butts (in spacing along of 33 ft. or 39 ft., as rooms are to widen to 7 yd. with 4-yd. ribs or to 8 yd. with 5-yd. ribs, respectively), and the retreat following is along the long break-line, and as extending both ways may have straight-diagonal or broken V-shape; the last named, however, is not so satisfactory. There are graphical

schemes in the study of the roof-effect in localities, as criteria of timber-cost and recovery of product, aimed to be more than 90 per cent. of the entire exhausted area. If control of roof-settlement is lost by failure to secure full breaks along the margin next the stumps of coal, there is over-running in dynamic strains, a squeeze will crush the coal down and a creep will raise up the soft bottom into the open space, and even the solid near by may be placed in stress if relieving roof-breaks are prevented by desultory working out of the areas. In some instances, the settlement of the roof has reached through 700 ft. of cover by fissuring to the surface. The exhaustion of panel-coal is intended to be independent of the removal of the entry-pillars, especially on the longer line of retreat. In the coke-region, with section-entries ("flats," as skirting along the level line of strike) having butt-entries 300 ft. apart with narrow rooms of 4 yd. or less between pillars of 14 yd. or more, on the rise side of the flat only, the removal of the thick ribs is attended by the withdrawal of the entry-pillars also, as one connected operation. So those mines are preferably developed to a boundary, and the recovery of the remainder of the coal from each area is postponed for the time.

A diagrammatic map on which the width of the regular belts composing the entries is exaggerated, but the spacing of the panels is shown in mean dimensions, is an ingenious device for indicating in detail the position of the permanent and temporary appliances for directing the ventilating air-currents and the lines for conducting power and other courses—as of prescribed foot-travel, etc.—in the wide-showing entries, with cut-throughs where the entries are parallel and near together. Although the working-areas are not shown in detail, yet the progress of exhaustion is shown plainly, so that at all stages the ventilation in broken territory may be regulated to allow the men to work in intake air, by the drawing off of the foul gases in goaves, directed to the return air-course from the older works in the rear of the progressing retreat-work.

Interference in the progress of the mine-workings is especially serious, as it inconsiderately affects the retreat-workings. Interruption, stopping and starting up on short notice, imposes irregularity on all the operating-conditions of the mine, and on the force of men as an adequate whole. It is contributory to

accidents, and catastrophes have had such ulterior cause. It ought to be remembered that "economy" is a word derivatively meaning "house-keeping." And used in its widest sense as applying to mine-management, economy means con-

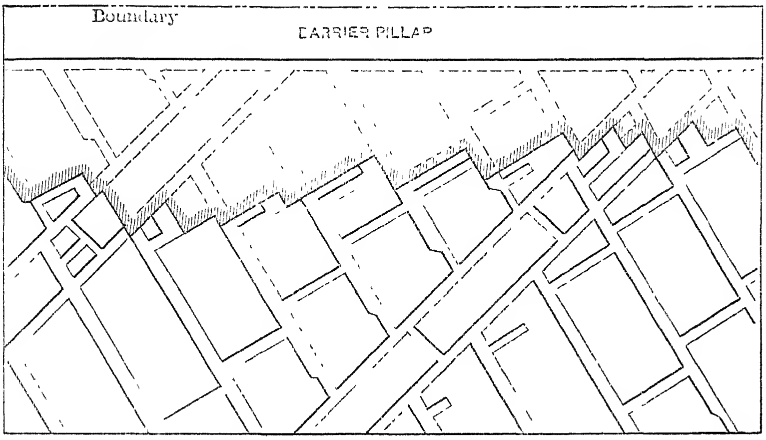
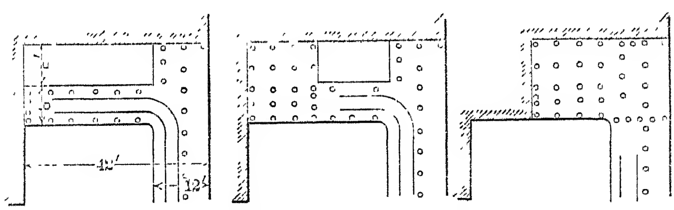


FIG. 1.—GENERAL PLAN OF RIB-DRAWING, CONNELLSVILLE REGION.

servation of living conditions so as to secure the welfare and best efforts of the men.

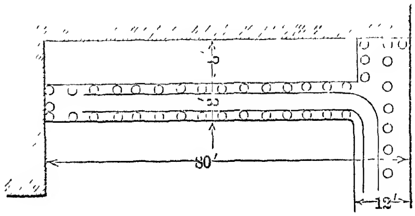
Of the accompanying illustrations, Figs. 1 to 4 show the methods of pillar-drawing followed in the Connellsville region. Fig. 1, from drawings by Edward McGrew, shows the method in general use with hand-power in this district, being that



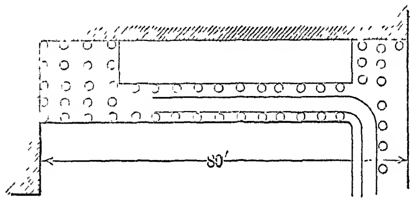
Rib cut across. Rib ready for first fall. Rib ready for last fall.

FIG. 2.—DETAILS OF RIB-DRAWING, CONNELLSVILLE REGION. ROOMS 42-FT. CENTERS.

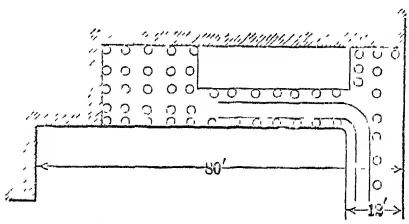
practiced by H. C. Frick Coke Co. and others; Figs. 2 and 3 give the details for rooms 42-ft. and 80-ft. centers; while Fig. 4 is the plan laid out by John H. Rayburn for the Thompson-Connellsville Coke Co., where compressed-air machines do undercutting and their aid is anticipated for initiating cross-



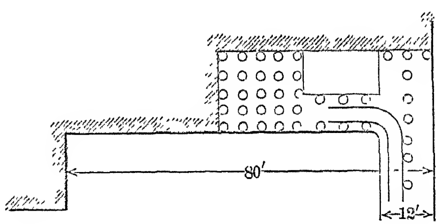
Rib cut across.



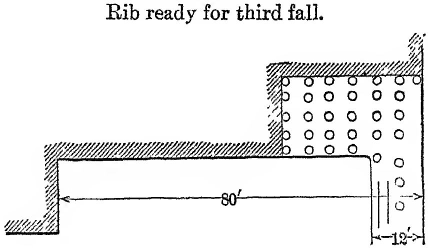
Rib ready for first fall.



Rib ready for second fall.



Rib ready for third fall.



Rib ready for last fall.

FIG. 3.—DETAILS OF RIB-DRAWING, CONNELLSVILLE REGION. ROOMS 80-FT. CENTERS.

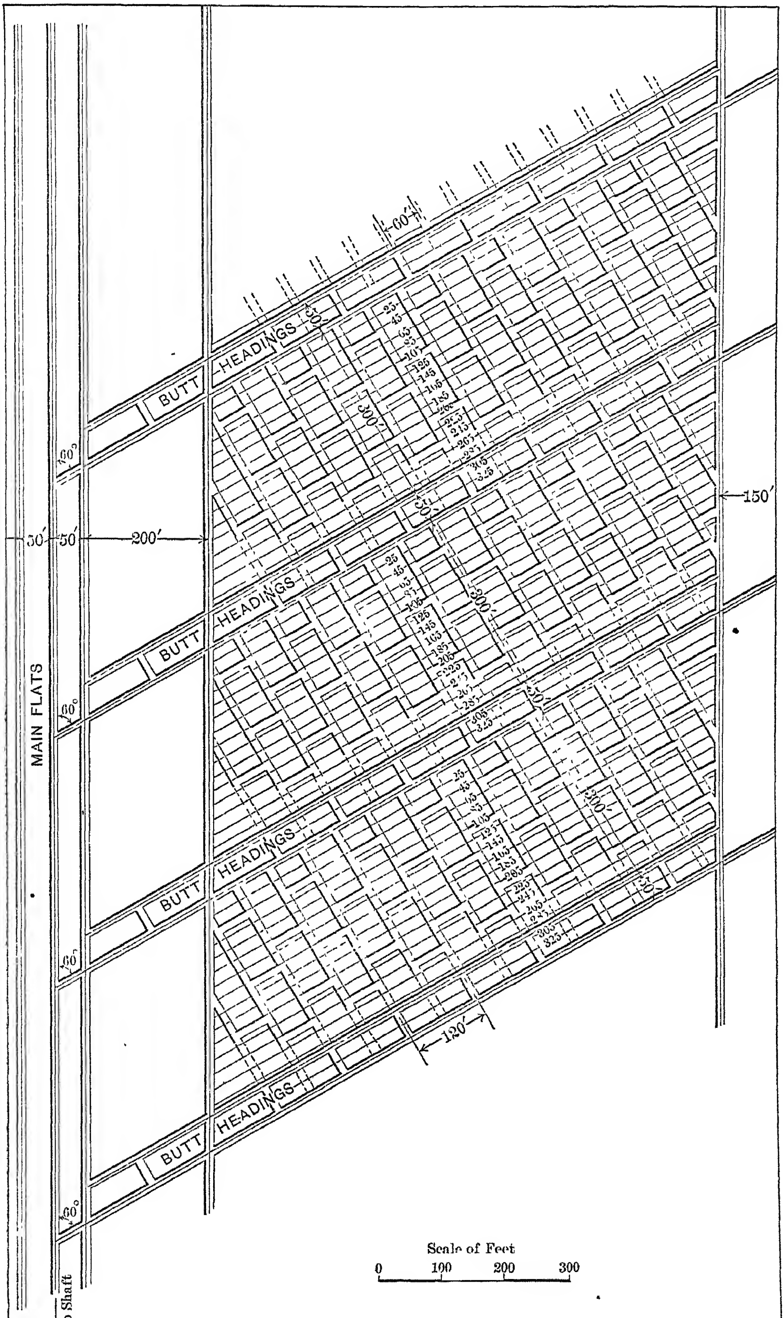


FIG. 4.—THOMPSON-CONNELLSVILLE COKE CO. METHOD OF RIB-DRAWING.

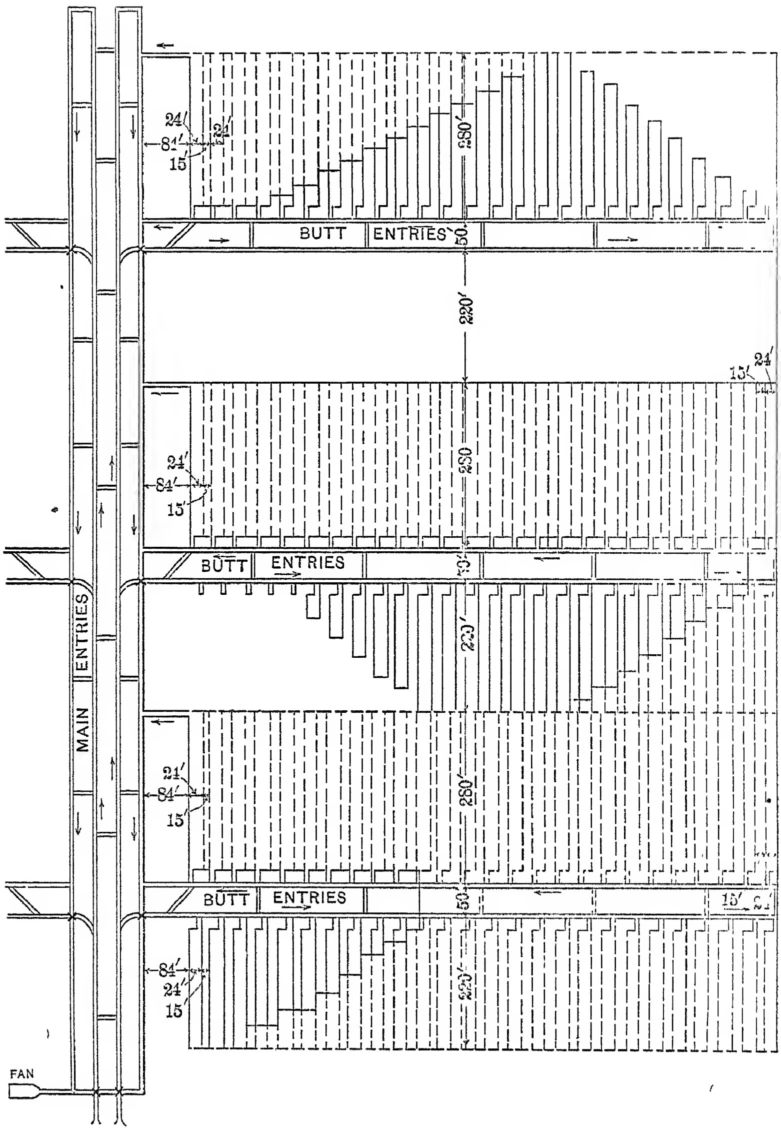


FIG. 5.—PITTSBURGH COAL CO. METHOD OF WORKING, PITTSBURG REGION.

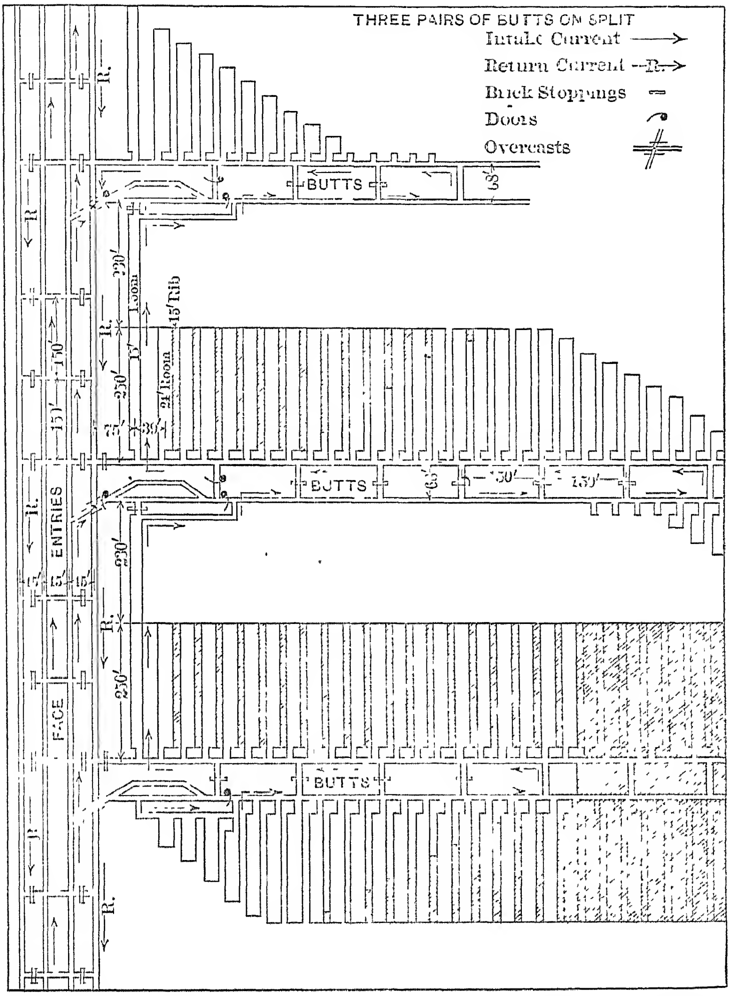


FIG. 6.—MONONGAHELA RIVER CONSOLIDATED COAL & COKE CO. METHOD OF WORKING, PITTSBURG REGION.

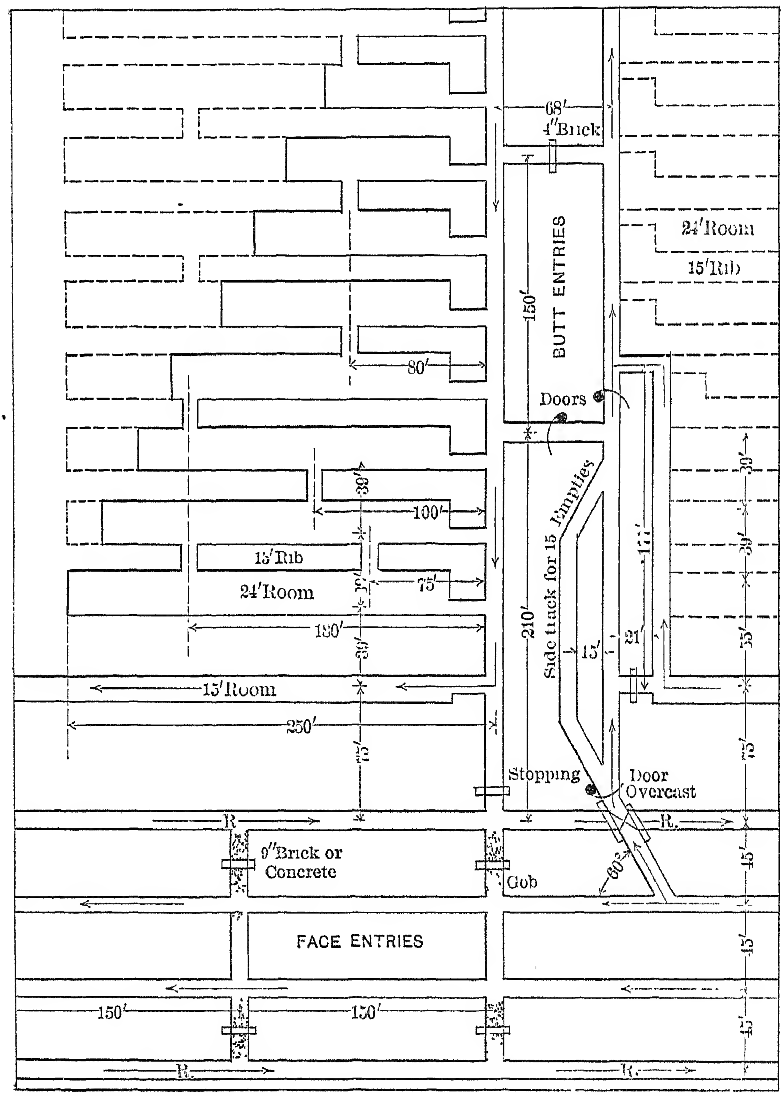


FIG. 7.—MONONGAHELA RIVER CONSOLIDATED COAL & COKE CO. METHOD OF WORKING, PITTSBURG REGION.

cut slicings, marked off regularly every 20 ft.; at least until the roof has taken weight.

The methods of working in the Pittsburg region are shown in Figs. 5 to 8. Of these, Fig. 5, furnished by the Pittsburgh Coal Co., is a composite plan of two types for control in rapid exhaustion across the panel. There are four face-entries. Rooms are 24 ft. wide, ribs 15 ft. Fig. 6, by the Monongahela River Consolidated Coal & Coke Co., is similar to Fig. 5 in showing together room-workings one side or both sides a pair of butt-entries at a time, as alternatives. And this company's Fig. 7 makes complete the showing of the consummate manner in

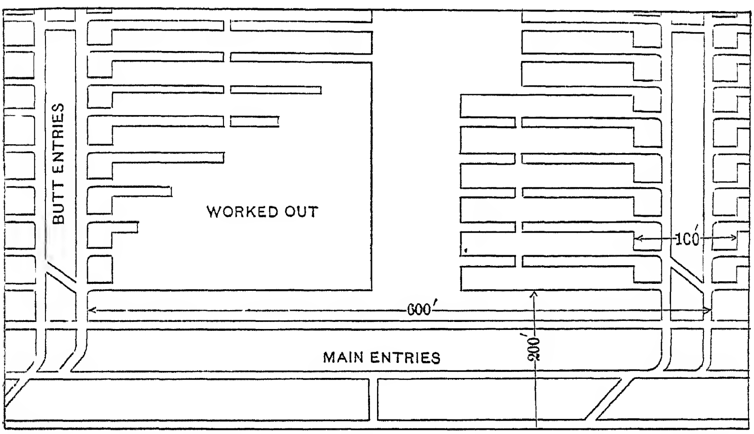


Fig. 8.—Method of Rib-Drawing, Pittsburg Region.

which the air-circulation is saved from interference by the transport.

Fig. 8, from a drawing by Edward McGrew, shows in good form the method in general use, as a proper succession of the stages of exhaustion. Everywhere the front of breaking roof is to be a line of faces *en echelon*, and its rear settled broken ground or a sufficient margin of solid coal.

It is to be noted that in the Connellsville region the coal is wanted fine for charring in the coke-ovens, while in the Pittsburg region lump coal is preferred for the market. The intermediate territory provides best for the by-product ovens, so called.

Dust-Explosions in Coal-Mines.

BY GEORGE S. RICE, PITTSBURG, PA.

(Pittsburg Meeting, March, 1910.)

THE extremely valuable papers and discussions on coal-dust explosions by Bache,[1] Eavenson,[2] Shurick,[3] Mannakee,[4] and Raymond [5] are of unusual interest to me, since it has been my duty to carry on investigations on this and allied subjects for the Technologic Branch of the United States Geological Survey during the past year, and to make a series of coal-dust tests in the explosion-gallery at Pittsburg during a short period when its use could be diverted from the pressing work of testing explosives.

In several of the discussions mentioned, it has been assumed that the Technologic Branch of the Survey was not paying attention to certain phases of the general problem, or to the specific question of temperature and humidity of mine-air in its effect upon the explosibility of coal-dust. This assumption is not correct, since from the time of the establishment of the "Mine-Accident Division," with its subdivisions of explosives, rescue-work, and mine-investigation, the mining engineers of the last-named section have made the subject one of chief consideration. Their general duties, including mine-sampling for government purposes, emergency-calls for rescue-work, and explosion investigations, have prevented undivided attention, but all of them have been supplied with instruments for such determinations, and special report-forms have been in use and a great many observations made in mining-districts to which the mining engineers have gone. With the idea of obtaining a uniform system of reporting these data, and in the hope that the work may be of service to mining engineers in the field, the collecting blanks in use are shown in Forms 1 and 2.

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[1] *Trans.*, xl., 667 to 673.
[2] *Idem*, 835 to 846.
[3] *Idem*, 846 to 847.
[4] *Idem*, 655 to 667.
[5] *Idem*, 807 to 809.

C. STATION HUMIDITY REPORT (1)

U. S. Geological Survey.

Date.... ....19........ Hour......................................
Mine..
Station.................................. ....
Distance from entrance or shaft..ft.
Dry bulb....................° Wet bulb...... ....° Barometer..................in.
Depression (t-t′)..............................° Relative humidity....................... ..... %
Passage width..................ft. Height....................ft. Area....................sq. ft.
Velocity air per min. reading........................ft. Corrected.......... ....ft.
Volume air per min....................... ....cu. ft.
Intake or return..
At station, is floor dry or moist?..
At station, are ribs dry or moist?....... ....
At station, is roof dry or moist?.. . ...
Method of sampling air...Sample No............. .

FORM 1—STATION HUMIDITY REPORT USED BY TECHNOLOGIC BRANCH OF THE U. S. GEOLOGICAL SURVEY.

D. GENERAL HUMIDITY REPORT (2)

U. S. Geological Survey.

General humidity condition of..Mine
Is the coal naturally dry or moist?..
Are the working places dry or moist?..
Are goaves dry or moist?.................. ....
In entries, is floor clean or dusty?.......... ....
Is dust pure or mixed with rock-dust or clay?.......... ....
In entries, is floor dry or moist?................ ....
Are ribs and roof coated with dust?..
Is the above dust dry or moist?..
Are timbers, if any, coated with dust?........... ....
Method of humidifying.......................... ....
(*a*) Sprinkling by cars..............................How often?...........................
(*b*) Sprinkling by hose..............................How often? ....
(*c*) Are ribs and roof sprinkled?.............................. ....
(*d*) Water sprays..Number..... ....
How located?... ....
(*e*) How many hours do sprays run?..
(*f*) Exhaust steam sprays..
Where located?...
(*g*) How far does steam fog the air?...
How much water, by any of above methods, is artificially introduced, in gallons per minute...................................per 24 hours....................................
General humidity notes:
....
....
....
....
....

FORM 2.—GENERAL HUMIDITY REPORT USED BY TECHNOLOGIC BRANCH OF THE U. S. GEOLOGICAL SURVEY.

Coal-dust and its treatment is a large subject, and one upon which hasty conclusions cannot be made. Nevertheless, some preliminary results of tests and observations will soon be published in a bulletin of the U. S. Geological Survey, under the title, Coal-Dust Problem and Proposed Remedies. This paper will include a summary of the experiments abroad and in the United States, and a discussion of the various methods for rendering coal-dust inert, together with the results of the preliminary tests made at the Pittsburg station and of observations made in coal-mines. The experiments have shown that moisture will render coal-dust inert, but the amount of moisture necessary for this effect is considerably in excess of what has hitherto been supposed. Mere dampening does not suffice. Moreover, it was found that the presence of air saturated with moisture is not, *per se*, a preventive of coal-dust explosions, as implied in Mr. Shurick's discussion, but the continued saturation of a ventilating-current does serve to moisten the dust, and if carried to a sufficient degree will render it inert to any ordinary initiating cause.

That explosions of coal-dust occur far more frequently in winter than in summer has been well established. This fact has led to considerable misapprehension that the temperature itself is responsible for explosive conditions, to overcome which it has been suggested that it would be sufficient to warm the intake air in winter. The important facts are that the return-air of a mine of any size is practically uniform in temperature, within a few degrees, winter and summer, and that it is generally in a nearly-saturated state, the relative humidity rarely falling below 90 per cent. Hence, it follows that where no artificial moistening is done, unless the air entering the mine carries the same amount of moisture that goes out of the mine in the return-air, over and above the drainage-losses, moisture is withdrawn to such an extent that the walls of the passage-ways and the coal-dust are more or less dried; the latter being generally left in an explosive condition during the winter period.

The observations and experiments mentioned indicate that the influence of barometric pressure, the temperature, and the condition of relative humidity of the atmosphere at the moment of testing is unimportant; the important feature being that otherwise inflammable coal-dust in the locality of the initiating

cause contains sufficient moisture at the time of exposure to concussion and flame to prevent its being raised and ignited. This percentage varies with the character of the dust, whether pure bituminous coal-dust or mixed with shale or slate, and the size and quantity of the dust-particles as well. The experiments show that all dry and pure or moderately pure bituminous and sub-bituminous dusts propagate an explosion under certain conditions.

In my opinion, Mr. Mannakee, in his paper on explosions in the Appalachian mines, lays undue stress upon the temperature, barometric condition, and humidity on the day of the explosion. The temperature for a considerable period prior to the explosion is of the utmost importance, but the experiments in the explosion-gallery have indicated that it has no discernible influence as compared with the condition and character of the dust itself. The gallery-tests at Pittsburg show that to render pure, finely-divided bituminous coal-dust inert when exposed to the flame of a blown-out shot, and the flame of the initial combustion, it must be sufficiently wet so that it can be molded in the hand.

While the dangers arising from improper use of explosives in mines have been fully recognized in the several discussions which have appeared in the *Transactions*, and while it is undoubtedly true that if black powder is properly used and the coal sheared and undercut, the greater number of the explosions resulting from this source would be eliminated, it by no means follows that all would be. The use or misuse of explosives is largely in the hands of the operators and miners. States may make laws, but unless the State inspectors are supported by the miners and the operators, the results sought will not be achieved. Not all States require the shearing and undercutting of the coal; and even when either is done, black powder, dynamite, or other long-flame explosives in gassy coals may cause fires or explosions. Under the conditions which prevail, and which are too deep seated to be changed by mere advice, the safest proposition is to use the explosives which have been tested by the U. S. Geological Survey in the presence of gas and coal-dust and found not to cause ignition by tests many times repeated, ands upplement this selection by efficient humidifying, spraying, or wetting throughout the mine.

Observations in the field by the mining engineers of the Survey have determined that the best and safest arrangement is

for all shots to be charged, tamped, and fired, with the use of electric detonators, by picked men, and not merely fired by shot-firers, who do not stop to inspect holes properly and who do not know just how they are charged. A further measure of safety, which would also improve the quality of the coal produced, would be to have the holes drilled by special drill-men. In other words, the use of explosives should not be put in the hands of ignorant, untrained men, or men who take dangerous chances. Speaking as one concerned in operating coal-mines for many years, my observation has been that more overcharged and dangerous shots are made by intelligent English-speaking men who are reckless, and sometimes criminally so, than by the inexperienced foreigners. I call to mind one explosion resulting from three overcharged shots prepared and fired by English-speaking miners, who, knowing these shots to be dangerous, purposely left long fuses, evidently expecting to get out of the mine, which was new and small, before the shots went off. Their plans were miscalculated, however, and the resultant explosion caused not only their own deaths, but the deaths of others as well.

The indirect effect of the misuse of explosives—namely, the weakening of the roof and blowing-out of props—is a most important factor in the large loss of life resulting from roof-falls. In my opinion, close supervision at the face, as pointed out by Mr. Saunders,[6] is most important. The great majority of underground foremen have too much ground to cover, and do not have opportunity to see that the directions about props are observed, nor, in fact, do they get to the face often enough to give such directions. If the Yorkshire system of deputies were adopted in the United States, I believe that there would be a great decrease in the number of accidents at the face. It is urged that the Yorkshire system is more expensive than that followed in the United States, since one deputy has only from 25 to 40 men under him. On the other hand, the decrease in accidents and the increased efficiency of the day or company men, who are under constant supervision, will, in my opinion, more than offset the increased cost; and even if it did not, the additional expense should willingly be paid by the consumer, from a humanitarian stand-point, as part of the cost of coal.

[6] The Conservation of Coal in the United States, *Trans.*, xl., 905.

The Fushun Colliery, South Manchuria.

BY WARDEN A. MOLLER, TIENTSIN, CHINA.

(Pittsburg Meeting, March, 1910.)

THE Fushun coal-field, now being opened up by the South Manchurian Railway Co., is connected with the main line by a branch, 30 miles long, from Sui Chia Tun, 10 miles south of Mukden, the capital of Manchuria.

The coal-field, which lies on the southern bank of the Hun river, 25 miles east of Mukden, consists of a long narrow valley, running east and west between hills composed of granitic and volcanic rocks. At the foot of the northern hills, the Hun river flows westward towards the Liao river, which drains the greater part of South Manchuria. At the foot of the southern hills, resting immediately upon volcanic rocks, and overlain by 20 ft. of alluvium, is a thick layer of Tertiary shales, carrying near the base two coal-seams. In many places the lower of these two seams is in immediate contact with volcanic lavas and basalts, the flows of which were evidently both prior and subsequent to the formation of the coal-seams. This contact has practically destroyed the lower seam for commercial purposes. The value of the field lies in the upper seam, which, for a length of nearly 8 miles, varies from 100 to 130 ft. thick. Of this seam, 85 per cent. is good coal.

The seam dips to the north conformably with the shales, at angles of from 25° to 30°, and appears to have an uninterrupted dip for about 1.5 miles before being possibly cut off by the northern hills.

The coal-bearing shales extend east and west for a length of about 20 miles, but towards the east the continuity of the seams is broken by intrusive rocks.

The coal is of the sub-bituminous variety, low in ash, bright in appearance, and breaks with a conchoidal fracture. It has the usual qualities of Tertiary coal—quick steam-raising and poor stacking qualities. Pieces of resin, varying in size from

a pinhead to a pea, are frequently found imbedded in the coal.

The following analyses have been made from samples of the thick seam:

Fixed Carbon.	Volatile Hydro-carbons.	Moisture.	Ash.	Sulphur.
Per Cent	Per Cent.	Per Cent	Per Cent.	Per Cent.
44 7	42.6	6.7	5.6	0.4
51.0	41.1	4.4	2.9	0.6
57.7	33.7	6.2	2.0	0.4

In practice, the quantity of ash and unburnt coal varies from 7 to 10 per cent.; the moisture-content of the freshly-mined coal is somewhat larger than that given above, owing to the delay between the picking of the sample and its analysis. Before the Russo-Japanese war, the field had been worked in only a desultory manner, due chiefly to official interference on account of the proximity to the Imperial Manchurian tombs. For many years both Coreans and Chinese have at different times unofficially mined coal for local consumption. The principal use of coal in Manchuria was for blacksmith's work, which required a higher class of fuel. Moreover, the Fushun coal could not compete with the semi-bituminous and coking-coals of the Yentai and Pen Hsi Hu fields.

At present the Fushun field has been proved to contain an enormous tonnage per acre comparatively near the surface, so that coal can be mined and sold at such a low price as to counteract the somewhat low economy of the fuel itself, especially since its low ash makes it a fair fuel for long locomotive-runs.

The present production, about 1,600 tons per day, is raised from a series of shallow pits, sunk during and subsequent to the Russo-Japanese war. The management, in order to increase the tonnage to 5,000 tons per day, is now sinking two pairs of shafts, of 18 and 20 ft. diameter respectively, a mile apart, and laying out surface-works and town-site on a corresponding scale.

The shafts most advanced, named the O'Yama, have up to date presented no difficulties; at present they are 370 ft. deep, and are entirely in green and gray shales, which have been proved by bore-holes to continue the entire depth. So far the shafts are practically dry, and it is expected that coal will be

reached at a depth of 1,200 ft. The shafts, lined with brick laid in cement, will probably be completed about the end of the year 1909.

The Toge shafts, 1 mile to the east, have just been commenced. The intention of the management is to provide all power, except for winding and ventilation, from a central electric power-plant, and already there have been installed in a capacious brick and steel power-house two 500-kw. Parsons turbine-driven dynamos, which supply three-phase current at 2,000 volts. Preparations are being made to erect two similar 1,000-kw. machines.

Steam is supplied to the engines around the O'Yama shafts by a battery of Babcock & Wilcox boilers, provided with a chain stoker and an automatic ash-remover; steam-pressure at 160 lb. per sq. in. is superheated 100° C., and feed-water for the boilers is passed through a set of Green's economizers.

The water for the town-site, boiler-consumption, and condensers is supplied from the river by an electric-driven turbine-pump, capable of raising 600 gal. per min. against a head of 150 feet.

The machine-, blacksmith-, and carpenter-shops, and air-compressor plant, are supplied with power from standard 25-kw. motors, which obviates the storage of a multitude of spare parts.

The town-site is being laid out on a liberal scale, and 50 or more two-story brick houses have already been erected for the higher class Japanese employees, including large offices, a school, a club, a company store, and free baths. The building of a hotel and a theater is in contemplation. The houses are heated by a central steam-plant, and are equipped with electric light, telephones, and hot and cold water.

The laboring class, which will probably amount in number to 3,000 Japanese and 10,000 Chinese, are housed in rows of houses suitable to each nationality.

The doubling of the present single-track main line between Sui Chia Tun and Tairen will be completed by the end of 1909. The ports for export will be Newchuang and Tairen, situated respectively 130 and 270 miles south of the coal-field. Tairen is about 15 miles northeast of Port Arthur.

The Combustion of Coal.

BY JOSEPH A. HOLMES, WASHINGTON, D. C., AND HENRY KREISINGER,* PITTSBURG, PA.

(Pittsburg Meeting, March, 1910.)

AT the Mining Experiment Station of the U. S. Geological Survey, in Pittsburg, an investigation of the process of combustion is being carried on in a specially-designed furnace having an unusually long combustion-chamber. This work is conducted by a committee consisting of H. Kreisinger, mechanical engineer; Dr. J. C. W. Frazer, chemist; and Dr. J. K. Clement, physicist. The problem, essentially one of physical chemistry, is extremely interesting to all who are concerned in the burning of coal or the construction of furnaces.

The main object of these experiments is to ascertain the relation between the amount of the volatile combustible driven from the coal and the combustion-spaces necessary to burn it completely. Our best steam-coals vary from 15 to 45 per cent. in volatile matter and from 40 to 75 per cent. of fixed combustible. The greater part of the latter is burned on the grate; but the volatile combustible leaves the freshly-charged coal and must be burned in the combustion-space. If this space is not large enough, the volatile combustible will leave the furnace only partly burned, and the result will be a considerable heat-loss and a smoky stack. Strictly speaking, the factor which determines the completeness of combustion of the volatile matter, after it has been mixed with a certain amount of air, is the length of time the mixture is allowed to remain in the combustion-space; but this length of time depends on the extent of the space itself. Let us suppose, for instance, that, when a given coal is burned at a certain standard rate, the volume of the volatile combustible driven off per second is 10 cu. ft. Adding, say, 10 cu. ft. of air, the volume of the resulting burning mixture is increased to 20 cu. ft. If the combustion-space is 20 cu. ft., the burning mixture will stay in it,

* U. S. Geological Survey.

on the average, 1 sec.; if 40 cu. ft., the mixture will remain, on the average, 2 sec.; whereas, if only 10 cu. ft., it can stay only half a second. It is therefore probable that in the burning of a given coal the completeness of combustion of the volatile combustible depends on the extent of the combustion-space. (For the sake of simplicity in the above illustration, the effect of temperature on the volume of the combustible mixture has not been considered.)

Effect of the Nature of Coal on the Extent of Combustion-Space Required.—Different steam-coals evolve different volumes of volatile combustible, even when burned at the same rate. Coal containing 45 per cent. of volatile matter evolves a much greater volume of gases and tar-vapors than that which contains only 15 per cent. Consequently, a furnace having a much larger combustion-space will be required to burn the former as completely as the latter.

The extent of combustion-space thus required depends not only on the volume of the combustible mixture, but also on its chemical composition. Thus, the volatile combustible of a low-volatile coal, when mixed with an equal volume of air, may require 1 sec. for complete combustion in a given space, while 2 sec. may be required to burn with the same completeness the same volume of the volatile combustible from a high-volatile coal. In other words, the combustion-space required to burn various kinds of coal is not directly proportional to the volatile matter contained in the coal.

Effect of the Rate of Combustion on the Extent of Combustion-Space Required.—With the same coal, the volume of the volatile combustible distilled from the fuel-bed per unit of time varies as the rate of combustion. Thus, when the rate of combustion is double that of the standard, the volume of gases and tar-vapors driven from the fuel is about doubled. To this increased volume of volatile combustible an equal volume of air must be added; and if the mixture is to be kept for only the same length of time within the combustion-space, that space should be about twice as large as for the standard rate of combustion. The space required for complete combustion varies, therefore, not only with the nature of the coal, but also with the rate of firing.

Effect of Air-Supply on the Extent of Combustion-Space Required.—Another factor which influences the extent of the combustion-space is the amount of air mixed with the volatile combustible. Perhaps within certain limits the combustion-space may be decreased when the supply of air is increased. On this point, however, anything said at present would be speculative. The facts must be determined experimentally.

Effect of Rate of Heating of Coal on the Extent of Combustion-Space Required.—There is still another and a very important factor influencing, for any given coal and any given air-supply, the extent of the combustion-space required—namely, the rate of heating of the coal when feeding it into the furnace. Our so-called "proximate" analysis of coal is indeed only very approximate. When the analysis shows, say, 40 per cent. of volatile matter and 45 per cent. of fixed carbon, it does not mean that the coal is actually composed of so much volatile matter and so much fixed carbon, but simply that at a certain rate of heating, given by certain standard laboratory-conditions, 40 per cent. of the coal has been driven off as "volatile matter." If the rate or method of heating were different, the amount of volatile matter driven off would also be different. Chemists can testify how difficult it is to get accurate checks on "proximate" analysis. To illustrate this point further, we may refer to the operation of the up-draft bituminous gas-producer. In the generator of such a producer, the tar-vapors leave the freshly-fired fuel, pass through the wet scrubber, and are finally separated by the tar-extractor as a black, pasty, semi-liquid substance. If this "tar" is subjected to the standard proximate analysis, it will be found that from 40 to 50 per cent. of it is fixed carbon, although the tar left the gas-generator as "volatile matter." The fact to be emphasized is, that different rates of heating of high-volatile coals will not only drive off different percentages of volatile matter, but that this material itself varies greatly in chemical composition and physical properties, especially as regards inflammability and rapidity of combustion. We may therefore conclude that the combustion-space required for complete oxidation of the volatile combustible depends, in some degree, on the method of charging the fuel; that is, on the rapidity with which the fresh fuel is heated.

In summary, it may be said that the combustion-space re-

quired to obtain a practically complete combustion depends on: (*a*) the nature of the coal; (*b*) the rate of combustion; (*c*) the supply of air; and (*d*) the rate of the heating of the fuel.

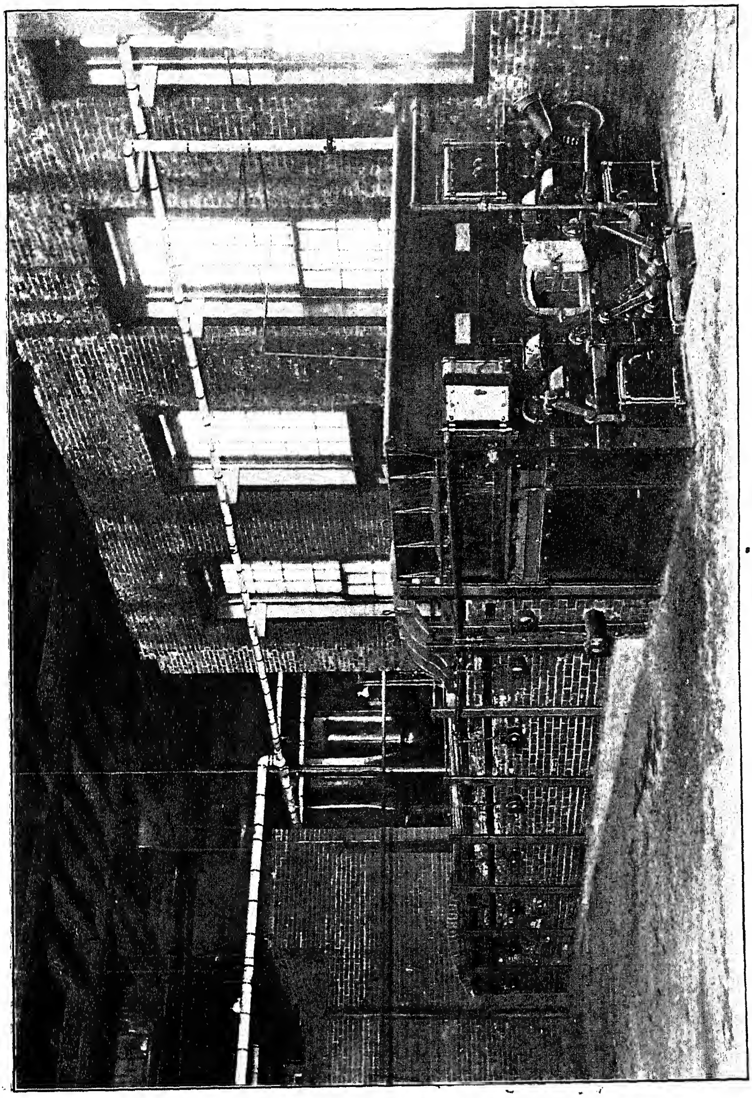

FIG. 1.—FURNACE WITH LONG COMBUSTION-CHAMBER.

To determine the influence of each of these factors is the object of the experiments under discussion.

Description of Furnace.—The furnace used in these investigations is fed with a standard Murphy mechanical stoker. The

special feature of the furnace is a large combustion-space, 3 by 3 ft. in cross-section and nearly 40 ft. long. To the volatile combustible leaving the fuel on the grate, air is added through

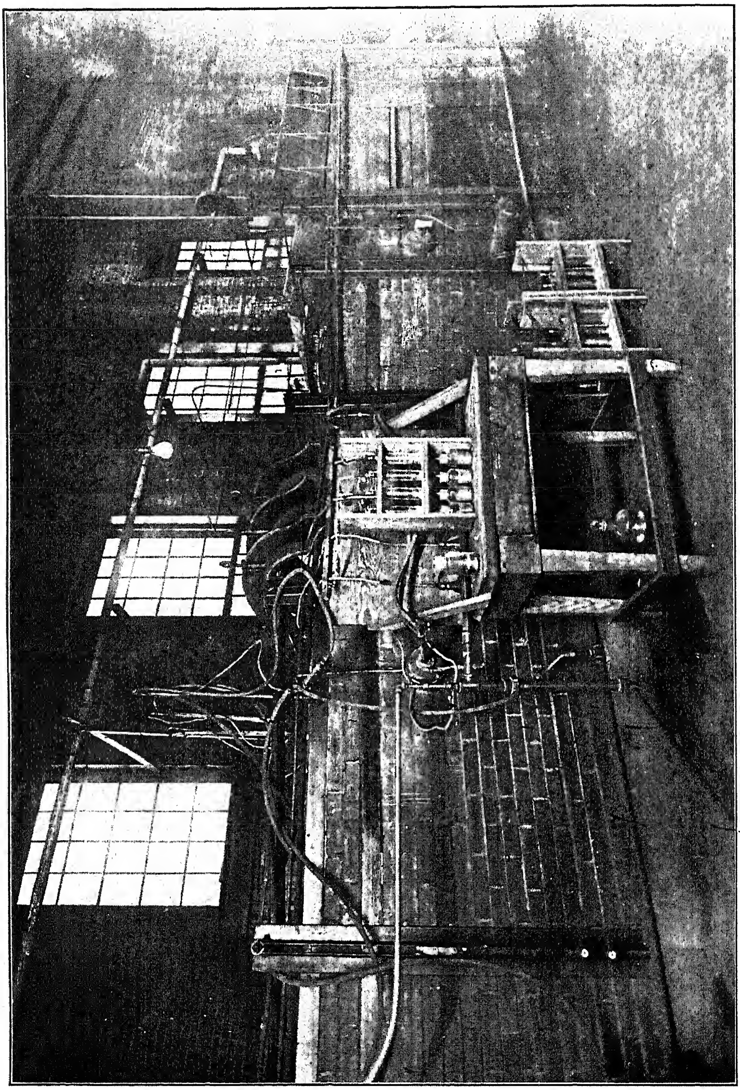

FIG. 2.—METHOD OF COLLECTING SAMPLES OF COMBUSTION-PRODUCTS.

numerous small openings above the grate; the mixture then passes through the combustion-space, and the products of combustion are discharged into the setting of a Heine boiler.

The difference of pressure necessary to move the gases through

the furnace is obtained by two fans, one of which exhausts the gases from the setting of the Heine boiler (into which they are discharged from the long combustion-space), while the other forces air under the grate of the stoker. During a test, the speed of the two fans is so adjusted that the pressure of the gases in the long combustion-chamber is slightly below that of the atmosphere, the object being to reduce to a minimum the leakage of air into the combustion-chamber, without making observation at the peep-holes disagreeable on account of jets of hot gases through them.

Figs. 1 and 2 show the general appearance and connections of the furnace. Fig. 2 indicates the methods of collecting the samples of gas. Three samples are being collected through the water-cooled tubes rising through and above the top of the furnace, as seen just to the right of the left window; and another through a similar tube entering the furnace through the observation-hole near and to the left of the table on which are the bottles in which the samples of gas are being collected.

Some of these water-cooled tubes contain three separate smaller tubes, each starting from the top, but ending at different distances; and it is therefore possible to collect at the same time three samples of gas through each main tube, or 12 samples in all, each coming from a different point in this cross-section of the furnace. And inasmuch as there are seven such cross-sections, and two additional observation-holes nearer the fire-box, it is possible, though perhaps never necessary, to collect 80 or 90 samples of gas during the same 20-min. period, each sample representing the average of the gases passing the open end of its small tube during this period.

Figs. 3 and 4 show longitudinal, horizontal and vertical sections of the furnace, and Figs. 5 and 6 show two typical cross-sections of the furnace. A Murphy mechanical stoker is at the front end of the furnace, and a 210-h.p. Heine safety boiler at the rear end. The observation-holes at 5-ft. intervals are shown through the left side-wall in Fig. 3.

The side-walls of this furnace are double, the inner being 9 in. thick and made entirely of fire-brick, while the outer is 8 in. thick and faced with red pressed-brick. Between the two is a 2-in. air-space. The arch forming the roof of the furnace is also double, the inner arch being of fire-brick, 9 in. thick, the

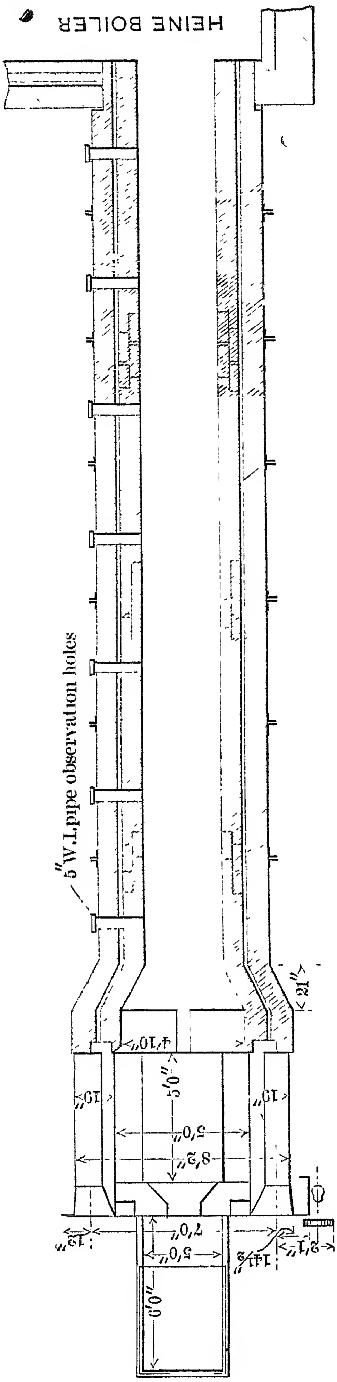

FIG. 3.—LONGITUDINAL HORIZONTAL SECTION OF THE FURNACE.

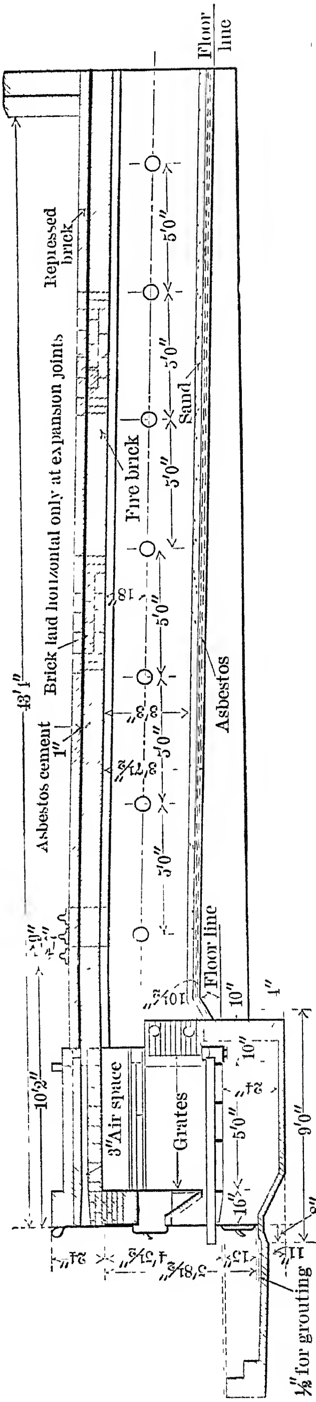

FIG. 4.—LONGITUDINAL VERTICAL SECTION OF THE FURNACE.

outer of pressed-brick and only 4 in. thick, and the space between the two being completely filled with a layer of asbestos 1 in. thick. The floor is made of a layer of asbestos-board laid

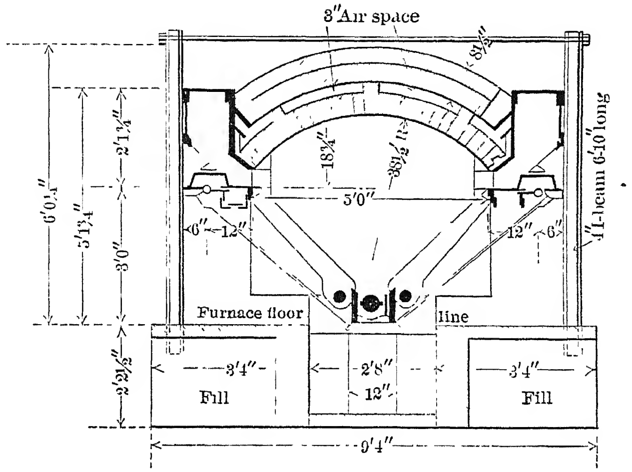

FIG. 5.—VERTICAL CROSS-SECTION THROUGH THE FIRE-BOX OF THE FURNACE, AND THE MURPHY STOKER.

directly on the earth, over which is spread a layer of sand, 3 in. thick, covered, in turn, with a 4-in. layer of fire-brick. In one of the side-walls there are eight peep-holes, 5 ft. apart, and

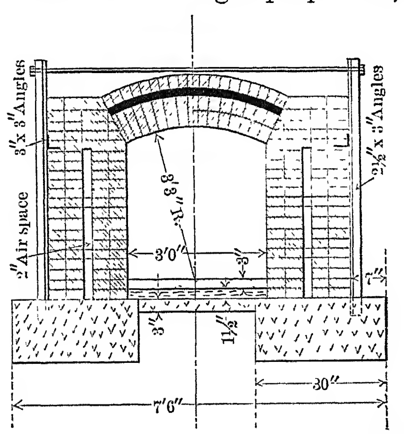

FIG. 6.—VERTICAL CROSS-SECTION THROUGH THE COMBUSTION-CHAMBER.

opposite each hole in the side-wall there are three smaller openings in the roof. Through these openings and peep-holes gas-samples are drawn for chemical analysis, and temperature-measurements and other necessary observations are made.

The Scope of the Experiments.—With the object explained in the preceding paragraphs, the following series of experiments has been planned:

Six or eight typical coals will be selected, each representing a certain group of coals of nearly the same chemical composition. Each of these coals will be investigated in a separate series of tests, the series consisting of several sets of tests, made with all conditions constant excepting the one, the effect of which on the size of the combustion-space is to be investigated. Thus, a set of four or five tests will be made, varying the rate of combustion from 20 to 80 lb. of coal per square foot of grate per hour, keeping the supply of air per pound of combustible and the rate of heating constant. This set of tests will show the effect of the rate of combustion on the extent of combustion-space required to obtain a practically complete combustion. Other variables, such as the composition of the coal, the supply of air, and the rate of heating, will remain constant.

Another set of four or five tests will be made with the same coal and at the same rate of combustion, but with a varying supply of air. This set of tests will be repeated for two or three different rates of combustion. Thus, each of these sets will give the effect of air-supply on the extent of required combustion-space, when the coal and the rate of combustion remain constant.

Still another set of tests should be made, in which the time of heating the coal while feeding it into the furnace is varied from 3 to 30 min. In each of the tests of this set, the rate of combustion and the air-supply would be kept constant, and the set would be repeated for two or three rates of combustion and two or three different supplies of air. These tests would give the effect of the rate of heating of fresh fuel on the combustion-space required to burn the distilled volatile combustible. These latter experiments will require a modification in the stoker-mechanism, and on that account may be put off until all the other tests on the selected typical coals are completed. As the investigation goes on, enough may be learned to permit the reduction of the number of tests in each series.

The chemical changes in the gases during the processes of combustion as these gases travel from the fire-box through the

elongated furnace, and the completeness of combustion in the successive cross-sections of the stream of gases, are determined mainly by the chemical analysis of samples of gases collected through the openings in the furnace at these points. The first of these cross-sections where gas-samples are collected passes through the middle of the bridge-wall; the others are placed at intervals of 5 ft. through the entire length of the furnace. Measurement of the temperatures of the gases and direct observation of the length and color of flames and of any visible smoke will be also made through the side peep-holes. These direct observations, together with the gas-analysis, will furnish enough data to determine the length of the travel of the combustible mixture before attaining practically complete combustion. In other words, these observations will determine the necessary combustion-space for various kinds of coal, burned under given conditions. Direct observations, and the analysis of gases at sections nearer to the stoker than the one at which the combustion is practically complete, will show how the process progresses towards completion. This information will be of extreme value in determining the loss of heat, due to incomplete combustion, caused by shortening the combustion-space.

Method of Collecting Gas-Samples.—The collection of gas-samples presents a difficult problem, when it is considered that the temperature of the gases in the furnace ranges from 2,400° to 3,200° F., and therefore that the samples must be collected by means of water-cooled tubes. About 25 preliminary tests have shown that the composition of the gases at the cross-sections near the stoker is not uniform, and that more than one sample must be taken from each cross-section. It has been decided to take nine samples from the cross-section immediately back of the stoker, and to reduce the number in the sections following, according to the uniformity of the gas-composition. Thus, about 35 simultaneous gas-samples must be taken for each test. These samples will be subjected to a complete analysis, and not merely to the usual determination of carbon monoxide, carbon dioxide, and oxygen.

It is also realized that some of the carbon-hydrogen compounds which, at the furnace-temperature, exist as heavy gases, are condensed to liquids and solids when cooled in the sampling-tubes, where they settle and tend to clog the tubes. To

neglect the presence of this form of combustible would introduce considerable errors in the determination of the completeness of combustion at any of the cross-sections. To meet this difficulty, special water-cooled sampling-tubes have been constructed and equipped with filters which separate the liquid and solid combustible from the gases. The contents of these filters also are subjected to complete analysis. To obtain quantitative data, a measured amount of gases must be drawn through these filtering sampling-tubes.

Measuring of Temperatures.—At present the only possible known method of measuring the temperature of the furnace-gases is by optical and radiation pyrometers. Platinum thermo-couples are soon destroyed by the corrosive action of the hot gases. The pyrometers used at present are the Wanner optical pyrometer and the Fèry radiation pyrometer.

The Flow of Heat Through Furnace-Walls.—An interesting side-investigation has been developed in the study of the loss of heat through the furnace-walls. As already stated, the side-walls of this experimental furnace contain a 2-in. air-space, which, in the roof, is replaced with a 1-in. layer of asbestos. To determine the relative resistance to heat-flow of the air-space and the asbestos layer, 20 thermo-couples were imbedded, in groups of four, to different depths at three places in the side-wall and at two places in the roof. In the side-wall, one of the thermo-couples of each group was placed in the inner wall near the inner surface; the second thermo-couple was placed in the same wall, but near the surface facing the air-space; the third thermo-couple was placed in the outer wall near the inner surface; and the fourth was placed near the outer surface in the outer wall. In the roof the thermo-couples were placed in the brick near the surfaces on each side of the asbestos layer. These thermo-couples have shown that the temperature-drop across the 2-in. air-space was much less than across the 1-in. layer of asbestos; in fact, that it was considerably less than the temperature-drop through the same thickness of the brick wall.

The results so far obtained indicate that, as far as heat-insulation is concerned, air-spaces in furnace-walls may be undesirable, and suggest that in furnace-construction a solid wall may be a better heat-insulator than a wall of the same total thick-

ness containing an air-space. If it is necessary to build a furnace-wall in two parts on account of unequal expansion, it may be wise to fill the space between the two walls with some solid, cheap, non-conducting material, such as ash, sand, or crushed brick. A more detailed account of these experiments will be given in a bulletin of the U. S. Bureau of Mines, entitled, The Flow of Heat Through Furnace-Walls.

Further experiments will be conducted concerning this phase of the subject. These combustion-investigations in the elongated furnace are being preceded and supplemented by chemical and microscopical investigations of the coals with a view to determining the origin and the hydrocarbon composition of each of the coals under examination.

A Commercial Fuel-Briquette Plant.

BY W. H. BLAUVELT, SYRACUSE, N. Y.

(Pittsburg Meeting, March, 1910.)

THE subject of fuel-briquetting has attracted much attention on the part of engineers and investors for the past 15 or 20 years, and especially in recent years, during which a number of plants have been built, with more or less commercial and technical success. Our technical literature contains numerous descriptions of certain processes, and discussions of the industry in general. For a history of the development of the art in the United States, and for descriptions of many of the processes which have been tried, reference may be made to the files of our *Transactions*, and, for more recent information, to a paper by C. T. Malcolmson, read before the First Annual Convention of the International Railway Fuel Association in June, 1909.

The purpose of the present paper is to describe the briquette-plant of the Solvay Process Co. and the Semet-Solvay Co. at Detroit, Mich., which, after a period of long and costly evolution, is now operating satisfactorily on a commercial basis. I believe that it will be of interest to some of our members to have the details of operation, power-consumption, labor, etc., as worked out on a practical and commercial scale in a going operation.

The briquette industry has had a somewhat checkered career in the United States, and much the larger portion of the failures has been due entirely to commercial causes. It seems axiomatic to say that before undertaking the installation of a plant for manufacturing briquettes it is necessary that the local situation should be carefully examined: First, to determine the kind of briquettes, if any, the local market will consume; that is to say, whether the available fuel-supply is such that briquettes would be a welcome addition, and whether the most promising field is the industrial or the domestic market. Second, if there is an available supply of raw material, as coal, coke, pitch, etc., which can be worked up into briquettes that will meet the market-conditions, and can be obtained at a price that will leave a margin for cost of manufacturing and a suitable profit. If this preliminary investigation does not result satisfactorily, the manufacturer should carry his plans to another place. Although these requirements are axioms of ordinary business prudence, yet neglect to observe them has been the cause of the failure of many undertakings, resulting often in the loss of the entire capital invested.

The next step is the selection of the process of manufacture best adapted to the materials at hand and the quality of briquettes to be produced. A number of different processes and briquetting-presses have been developed, both in Europe and in the United States. In Continental Europe the manufacture of briquettes has reached large proportions, and has become an important industry. The usual shape of the European briquettes is that of a rectangular block with rounded edges, weighing from 10 to 0.5 kg., or less. The larger sizes, which are usually broken up before firing, have the advantage of greater convenience in storing. For household purposes the housewife patiently breaks up the large briquettes with a hammer, but not having been accustomed to prepared sizes of anthracite, the annoyance, dirt, and waste caused thereby are not realized. The smaller briquettes, usually of the eggette form, are largely used, especially in Belgium and parts of France.

There is a sharp line in manufacturing between the eggette and other forms. The eggettes are made on rotary presses, the coal and binder being pressed between the faces of two broad wheels or rolls, having faces which are cupped out to

receive and compress the briquettes into form. Generally speaking, it is not practicable on these rotary presses to produce briquettes larger than about 6 oz. in weight. On the other hand, the capacity of the reciprocating presses, which are used in making the larger briquettes, falls off rapidly as the size decreases, and it is usually not commercially economical on these presses to make briquettes which weigh much less than 1 lb. each. Some forms of press have been developed, notably in the United States, which, in a measure, combine the features of the rotary and reciprocating presses, and which would produce briquettes of intermediate size between the European rotary and the reciprocating press. These machines, however, have not been fully developed to commercial success.

In planning for the Detroit plant the domestic market appeared the most promising, and therefore a rotary type of press was selected which would produce small briquettes, or eggettes, each of about 2 oz. weight. After much experiment, in which it was learned that a good many things must not be done and that a few points were absolutely essential to the success of the operation, the plant was developed as it now stands.

The raw materials used are coke-breeze from an adjacent coke-oven plant, and dry non-caking coal from either the Hocking Valley or the Jackson Hill district. These materials are used in equal proportions, by weight, mixed with from 8 to 9 per cent. of the binder. Coke-breeze appears to require somewhat more binder than coal; perhaps on account of its porosity, which allows the binder to be absorbed into the cell structure without useful effect. Probably further experience will result in reducing the percentage of binder used. Coal-tar pitch was selected as the binder, since experience has shown that, broadly speaking, it is the only thoroughly successful binder, generally obtainable at a moderate cost, which can produce a water-proof briquette.

Two processes are in general use, one using soft pitch, the other hard pitch. In the former the pitch is melted by steam-heat or in an oven fired with coal or other fuel, and mixed with the solid fuel while liquid, while in the latter process the pitch is made hard enough to permit of its being ground to a fine powder. In practice this latter means that, adopting the usual tar-distiller's test, the pitch should be somewhat

harder than can be chewed in the mouth without fracture after the pitch has been permitted to reach the temperature of the mouth. Unfortunately, no more scientific method of testing the hardness of pitch has been agreed upon as yet. The whole subject is in a rather chaotic stage, so the old tar-distiller's chewing-test is still largely employed.

The hard-pitch method was selected for the Detroit plant, partly because the soft-pitch methods are always a source of danger from fire, but mainly because it is believed that the hard-pitch method is the one which should be developed, with a view to obtaining the pitch at the lowest price delivered at the briquette-plant. Hard pitch is more easily transported, either in very cheap packages, or, in cold weather, in bulk, and it can be produced at lower cost, since a larger portion of the more valuable oils contained in the original tar have been removed. Possibly the hard-pitch method requires a slightly larger quantity of binder with some coals, but the lower cost at which it can be manufactured more than offsets this objection, and it also has the advantage of producing less smoke in combustion.

The accompanying diagram, Fig. 1, shows the arrangement of the different parts of the plant.

The process as at present conducted is essentially as follows: The coal is unloaded into a track hopper and elevated to the coal-bin. As slack coal is always used, no preliminary crushing is necessary. The coke-breeze usually contains considerable moisture, so in order to maintain constant conditions the breeze is dried in a rotary drier before being elevated to its storage-bin. The coal and coke are brought together from the bins into a measuring-machine, Fig. 2, which delivers the two materials in any desired proportion to a hammer-mill of the Jeffrey or Williams type, Fig. 3, in which the mixture is finely ground. The pitch, which meets the coal and coke in this mill, is brought from the pitch-storage shed by a pitch-man, who breaks it into pieces of convenient size for feeding to the pitch-cracker. This cracker consists of a pair of rolls, 16 in. in diameter and 12 in. face, with small V-shaped corrugations, running at a speed of 85 rev. per min. These rolls crush the pitch to about $\frac{3}{8}$ in. in diameter, and give no trouble unless the pitch employed is too soft to retain its brittleness at the temperature of the atmosphere. The crushed pitch falls from the rolls into a small hopper,

from which a screw-conveyor, which acts as a measuring-machine, delivers a definite quantity of pitch on to a small belt-conveyor, which in turn discharges the pitch into the mouth of the hammer-mill, as above stated.

By grinding the coal, coke, and pitch all together in the hammer-mill the three materials are thoroughly mixed, and opportunity is given for each particle of the coal and coke to receive a coating of pitch powder, which, of course, is the theoretical condition to be striven for. Practice has shown that in pulverizing coke-breeze the wear on the hammers of the mill is too great, and it is planned to install a small pair of chilled-iron rolls for pulverizing the coke, leaving the hammer-mill for the coal and pitch. The pulverized coke, coal, and pitch will be delivered direct to an elevator-boot, which raises the mixture to the top of the building. From this elevator the mixture is conveyed in a rotary conveyor, in which the different ingredients are thoroughly mixed, to the rotary heater, Fig. 4, consisting of a cylinder, 40 in. in diameter and 21 ft. long, set at a slight inclination. By the revolution of this cylinder the mixture passes from the upper to the lower end and is thoroughly mixed thereby. The cylinder is also heated to a temperature of about 80° C. by superheated steam introduced through a perforated pipe extending the length of the cylinder. This steam is superheated by the waste heat from the coke-breeze drier. In this manner the several components are accurately and uniformly proportioned, pulverized, mixed, and heated to a controlled temperature; also, a small quantity of moisture is introduced, which is desirable to insure a pliant and readily moldable material for the press. In some processes a vertical cylinder with stirrers is employed instead of a horizontal rotating cylinder, the mix being heated with superheated steam in a similar way.

From the mixing-cylinder the material is delivered by a short chute to the feed-box over the rotary press. The press used at this plant, Fig. 5, is of French manufacture and of the standard Belgian type. It consists of a heavy frame supporting two rolls, each of which carries in the middle a gear having 7-in. face and 2.5-in. pitch, and on each side of the gear a steel tire, 5.25 in. wide on the face by 2.5 in. thick. These gears and tires are accurately fitted on to the drum or roll, and keyed

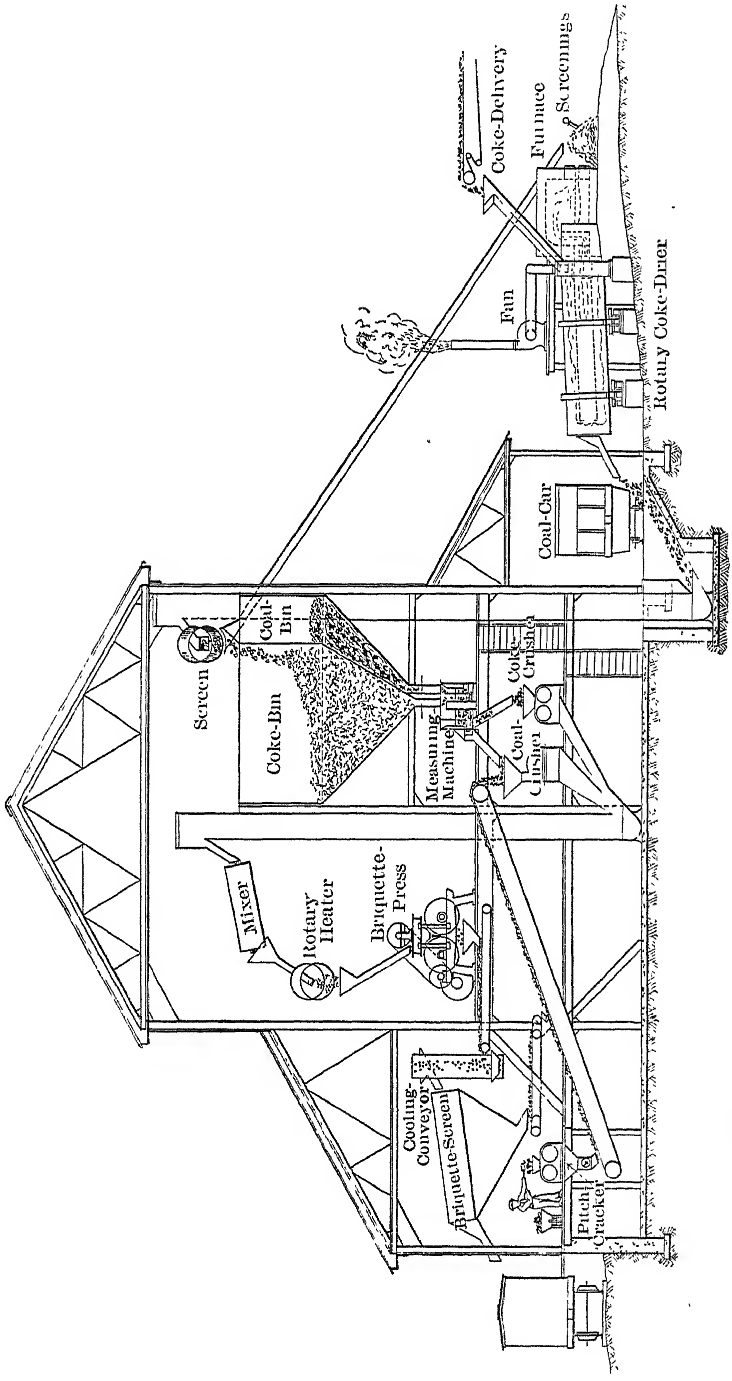

FIG. 1.—DIAGRAM SHOWING THE ARRANGEMENT OF THE FUEL-BRIQUETTE PLANT OF THE SOLVAY PROCESS CO. AND THE SEMET-SOLVAY CO., DETROIT, MICH.

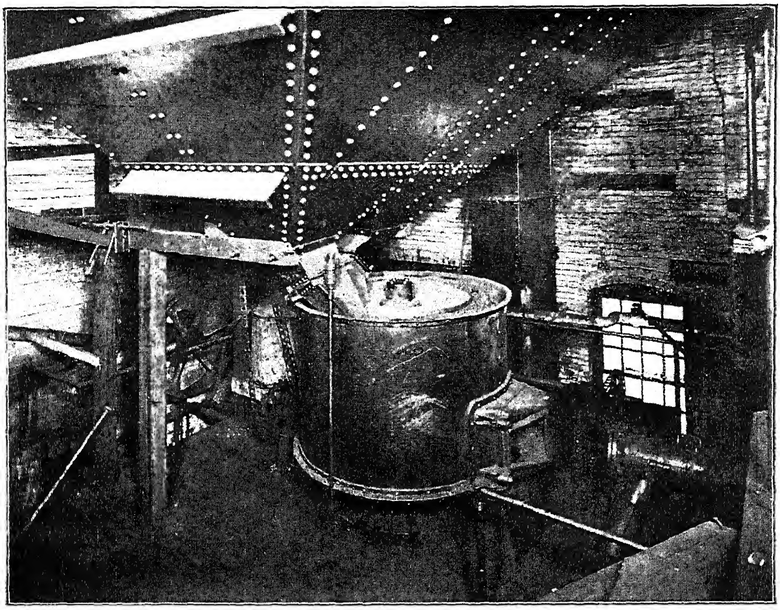

FIG. 2.—STORAGE-BINS AND MACHINE FOR DELIVERING MEASURED PROPORTIONS OF COAL AND COKE.

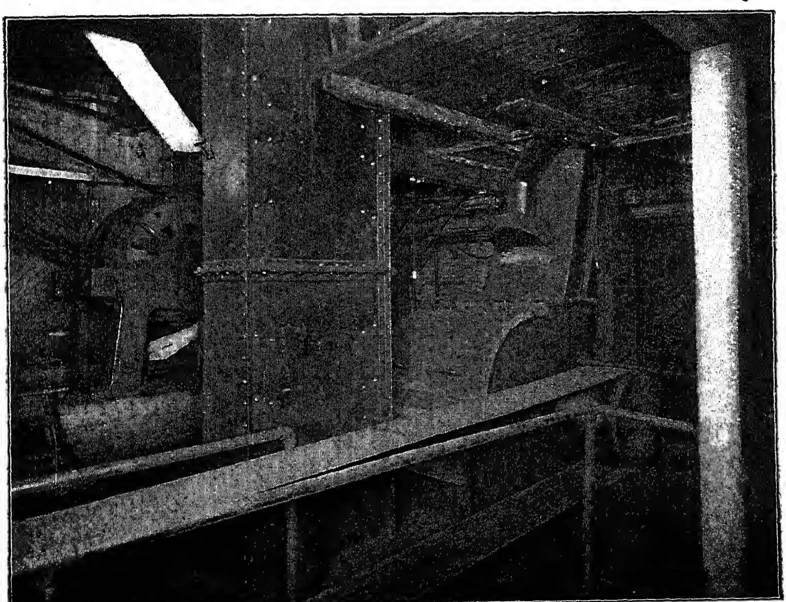

FIG. 3.—HAMMER-MILL FOR PULVERIZING THE COAL AND COKE.

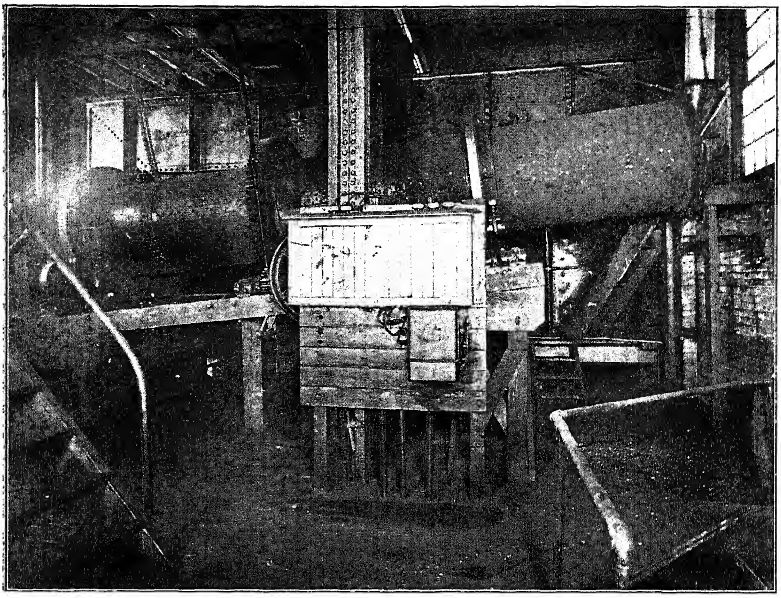

Fig. 4.—Rotary Heater and Mixer. The Superheated Steam Enters at the Left-Hand End.

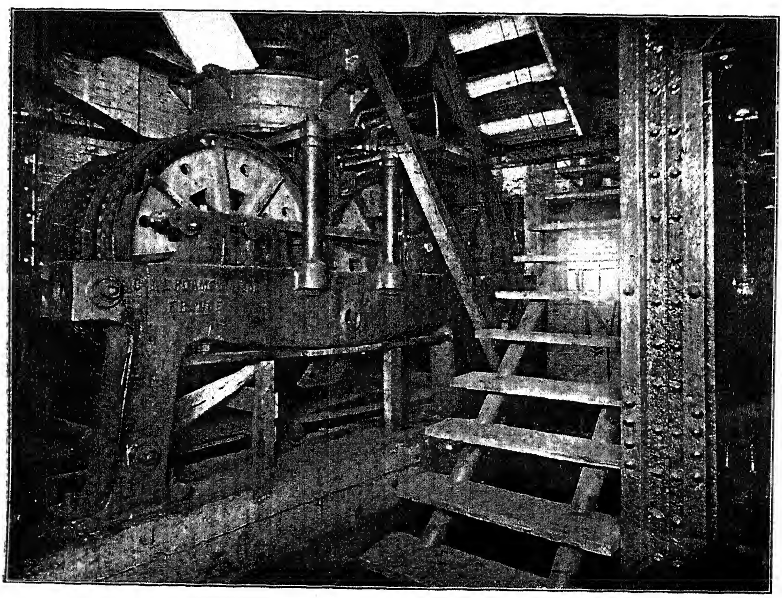

Fig. 5.—Briquette-Press. The Feed-Control Levers to One Pair of Rolls Are Shown Directly Under the Feed-Box.

and bolted to place. The faces of the tires are hollowed out or cupped into recesses about 2.25 in. long and $1\frac{7}{8}$ in. wide. There are three rows of these cups, arranged as closely as possible, so that each tire contains 207 cups. The rolls are accurately registered, so that the cuppings in the adjacent tires shall produce the egg-shaped briquette, the shape and dimensions of the cuppings determining the form and weight of the briquettes. Provision is made for turning up the faces of these steel tires as they wear and for cutting out the cups, so that the life of the tires may be thereby prolonged. It is found that the wear is much greater with coke-breeze than with coal alone, and it is proposed to substitute rolls with a chilled-iron face in place of the steel. It has been found in our experiments that even the alloy-steels, especially designed to resist wear, are not as satisfactory as chilled cast-iron. The cutting-action of the sharp particles of coke wears the best manganese-steel, for example, perhaps even more rapidly than ordinary tool-steel.

Adjustments are provided on the frame of the press which allow the rolls to be brought together as the tires wear or are dressed down. The rolls are driven by means of a pinion engaging one of the pair of gears. The pinion-shaft in turn is driven through gearing by a belted motor. The belt affords a convenient safeguard in case of foreign material getting between the rolls and blocking them. If a direct drive is used, carefully-designed breaking-pins are essential.

The mix which has been delivered into the feed-box is constantly stirred therein by rotating paddles, and fed down to the rolls through two chutes, specially arranged with sides which may be moved in parallel, thereby controlling the feed of the mix to the rolls. It is essential to the production of uniform briquettes that, in addition to the mix being uniform in composition and temperature, the quantity delivered between the rolls be accurately controlled, since it is the resistance of the material itself which determines the amount of pressure under which the briquette is produced.

From the press the briquettes fall on a conveyor-belt and are delivered to a cooling-conveyor outside of the building. This cooling-conveyor is a slow-moving rubber belt of sufficient length to permit the briquettes to cool sufficiently for the

pitch to harden and give strength to the briquette to resist breakage as it passes through the screen and falls into the railroad-car. At some plants the cooling-conveyor is constructed of a woven-wire belt, which would seem to be advantageous in that it gives greater access of air to all sides of the briquettes. When the briquettes have been sufficiently cooled to have acquired the necessary toughness, they are delivered from the cooling-conveyor to a rotary screen, which separates out any fine material and knocks off any fins which may adhere to the briquette on account of wear of the rolls. From the rotary screen the briquettes fall into railroad-cars for shipment, or into storage-piles. A small-sized conveyor returns the fine unbriquetted material and the fins to the plant and delivers it back to the hammer-mill, so that it is pulverized and mixed thoroughly with the new material.

The capacity of this plant has not yet been fully developed. It has been brought up to 9 tons per hour, and it may reach 10 tons. The main features of the plant are, however, arranged to have a capacity equal to the output of two presses, and it is the purpose, as the market develops, to add the second press. This addition would result in a very material reduction in the cost of operation, since the labor required will be but little more than at present, about two more men. The power required for operating the elevators, conveyors, etc., and the repairs on these parts of the plant will be but little more than at present when the tonnage has been doubled. Probably the power required and the wear on the pulverizing-mills, presses, and some other parts will be more nearly in proportion to the tonnage. An examination of the costs given below, based on the output of one press and two presses, shows very clearly the effect of output on the cost of briquette manufacture.

Repeated readings have been made of the power consumed in the different parts of the plant, with the following average result:

Motor Driving.	Brake Horse-Power.
Breeze-conveyor to drier,	1.50
Breeze-drier and ventilating-fan,	2.85
Pulverizing-mill,	22.00
Elevator-shafting and rotary mixer,	10.00
Briquetting-press,	25.00
Total,	61.35

The press-motor also drives the cooling-conveyor, rotary screen, pitch-cracker, and conveyor.

The consumption of steam in the rotary heater and mixer depends partly upon the temperature of the atmosphere and partly upon the quantity of binder and its melting-point. Tests at the Detroit plant, extending over a number of days, show a consumption of 206 lb. of steam per ton of briquettes produced, during a period of somewhat slow operation. No tests have been made during a period of steady running at full speed, but the above steam-consumption per ton of product would undoubtedly be decreased by a larger output. The superheating of the steam is desirable in order to maintain it in a dry state, and prevent the addition of too much water to the mix. The experiment was tried of heating the outside of the rotary mixer and heater with a gas-flame, but the use of superheated steam seems to be more satisfactory.

The labor on a plant similar to the one described is approximately:

	Cost Per Hour
1 foreman,	$0.50
1 press-man,	0.26
1 oiler, breeze-drier- and conveyor-man,	0.18
1 pitch-man,	0.18
1 briquette-loader,	0.19
2 laborers, at 17 cents,	0.34
Total,	$1.65

When producing 9 tons of briquettes per hour, the labor-cost is 18.3 cents. Two presses would double the output, but would only require two more men at 18 cents, and a second press-man at 26 cents per hour, which would bring the labor-cost to 12.6 cents per ton.

The use of coke-breeze increases the wear on all apparatus with which it comes in contact, but most especially on the briquetting-rolls. The question to be determined in each case is whether the breeze can be obtained at a cost sufficiently low in comparison with coal to offset the increased repairs due to its use. In addition, coke-breeze has the advantage of being smokeless, so that its use reduces the smokiness of soft-coal briquettes. The difference in cost of maintaining the briquetting-rolls with and without coke-breeze is so great that

two cost-sheets are given below, which will illustrate the approximate cost of manufacturing briquettes under these two circumstances in a plant similar to that here described, producing 9 tons of briquettes per hour. If the plant were operated on the basis of the larger production obtained from two presses, it would be safe to estimate a reduction in these two costs of, say, 10 cents per ton.

	Cost Per Ton of Product.	
	Using 50 Per Cent. of Breeze.	Using 100 Per Cent. of Coal.
Labor,	$0.183	$0.183
Power, at 1.25 cents per kw-hr.,	0.072	0.072
Steam, 206 lb., at 0.5 cent per h-p-hr.,	0.034	0.034
Breeze-drier and superheater fuel,	0.03	0.011
Miscellaneous supplies, oil, waste, lights, and water,	0.03	0.03
Repairs on rolls,	0.191	0.035
Other repairs,	0.05	0.035
Total,	$0.60	$0.40

To obtain the total operating-cost of a plant such as has been described, and operating under similar conditions, the cost of the pitch for binder and of the coal, coke-breeze, or other fuel used, must be added to these figures.

In order to have an illustration of such costs, let us assume that suitable slack coal can be obtained at $2, coke-breeze at $1, and pitch at $8 per ton, delivered at the plant. Since, as above pointed out, the use of breeze requires somewhat more binder, we will assume the use of 7.5 per cent. of binder with the coal and 9 per cent. with the mixture of coal and breeze.

	Cost of One Ton of Briquettes.	
	Equal Parts of Coal and Breeze.	All Coal.
0.455 ton coal, at $2,	$0.91	
0.925 ton coal, at $2,	......	$1.85
0.455 ton breeze, at $1,	0.455	
9 per cent. of pitch, at $8,	0.72	
7.5 per cent. of pitch, at $8,	......	0.60
Cost of briquetting, as above,	0.60	0.40
Total,	$2.685	$2.85

Many coals can be briquetted with less than the above quantities of pitch, and, of course, the costs of coal and coke-breeze vary at every plant, but the substitution of local prices in the above tables will give approximately the cost of briquettes in

a plant having about the capacity of the one described. In larger plants the cost of operation is less, and with briquettes of large size for industrial purposes the cost would be materially reduced.

In contemplating the erection of a plant, not only must a press be selected suited to the material to be treated and also to the kind of briquette demanded by the available market, but it must be remembered that a briquette-plant includes much more than the mere press. To insure economical operation, the preparation and handling of the material must be carefully studied and planned for, and the whole combination of driers, bins, elevators, measuring-machines, crushers, mixers, heaters, conveyors, etc., must all be combined with the press into a single effective and economical unit, which will insure a uniform proportioning of the materials, their thorough and intimate mixture, their heating to a sufficient and uniform temperature, and their delivery to and removal from the press at a constant and reliable rate. If the various units entering into a briquette-plant are thus successfully combined with careful economy of power and convenient arrangement, so that the labor is maintained at a minimum, a briquette-plant should be a reliable and assured source of profit in locations where a satisfactory raw material and a market for the product obtain. While the processes are simple and easily understood, it is essential that the different conditions be rigidly maintained, and usually it is only after the press-man and other more important operatives have attained certain instinctive recognition of the conditions that the plant becomes thoroughly successful.

The Combustion-Temperature of Carbon and Its Relation to Blast-Furnace Operation.

BY CLARENCE P. LINVILLE,* STATE COLLEGE, PA.

(Pittsburg Meeting, March, 1910.)

It is recognized that, in all metallurgical operations, the greatest possible uniformity in all conditions is essential to the best results. It is the constant aim of metallurgists to secure this uniformity, and their success in this respect has been in many cases the cause of great advances in the metallurgical processes themselves.

Of all such processes, the production of pig-iron in the modern blast-furnace is perhaps the most complex and the most difficult of control. Notwithstanding the great progress of recent years, there is still great difficulty in obtaining anything like uniform conditions and results in the manufacture of pig-iron. The reason is not far to seek. The ores vary in composition and physical properties, often carrying a varying amount of gangue material, unevenly distributed. The coke may vary in composition, cell-structure, moisture, hardness, and other properties. The blast, the weight of which far exceeds that of any other constituent, varies in its moisture-content and consequently in its chemical composition; and its temperature, by reason of the method of heating, is not uniform, but, besides varying from time to time with atmospheric and other outside conditions, has a maximum and minimum for every changing of stoves.

Now, with regard to these variations, the modern demand for uniformity has been greatly aided. Chemical analysis has made it possible to select ores so that the composition of the charge can be kept fairly constant; coal may be obtained from a given field, and coked in the same kind of ovens under conditions which make the fuel practically uniform; the Gayley

* Associate Professor of Metallurgy, School of Mines and Metallurgy, the Pennsylvania State College.

dry-blast process has rendered the composition of the air almost constant, by reducing the amount of moisture to a constant quantity; and the temperature of the blast is often maintained almost constant, either by the use of "equalizers," or by the admixture of cold air with the hot blast.

An important point, however, which indeed has not been entirely overlooked, but which has not received the consideration it deserves, is the hearth-temperature. It would seem to be a matter of the greatest importance to maintain the temperature of the blast-furnace hearth at a uniform point, if possible. Many furnace-troubles may be traced to a changing hearth-temperature, and if it were possible to maintain this constant, it is probable that these troubles would be greatly diminished.

The temperature of the hearth is determined by several factors, notably: the temperature of the burning coke; the amount of material to be fused; the melting-points of the materials fused; the reactions other than carbon-combustion; and irregularities, such as slips, which cool the hearth by precipitating cold material into it. Of these, the variables are the combustion-temperatures and the irregularities due to slips. It is probable that if the variations of the first of these could be overcome, the effect of the latter would be greatly diminished. Combustion-temperature, then, is the important consideration. The higher the combustion-temperature, other things being equal, the higher will be the temperature of the hearth, and the more rapid will be the rate of fusion of iron and slag.

The lower portion of our iron blast-furnace may well be considered as having several zones. In the lower part of the hearth is found the melted pig-iron, covered with a thick layer of melted slag. Once the pig-iron has reached the hearth, there is little chance for any further change in it. Its temperature may be increased through contact with the slightly hotter slag, but this is partly counteracted by conduction to the ground and the walls of the hearth. The slag, being in direct contact with the combustion-zone, has a much greater opportunity to become superheated, and is only deterred from reaching a temperature approximating the combustion-temperature by the fresh material constantly melting and falling into it. The hottest part of the furnace is undoubtedly just in front of and above the tuyeres, where the coke is burned. The hot

gases, as they rise, give up their heat rapidly to the descending materials.

At a greater or less distance above this place will be found the fusion-zones, first for slag, and higher for pig-iron. This distance will depend in each case upon the temperature of the combustion-zone, the melting-points of the materials, and the rapidity with which they pass downward through the furnace.

It can be readily seen that, given the proper temperature, and the amount of slag and iron to be melted, an increase in the amount of coke can only result in delaying the operation of the furnace, since this coke has to be burned out before the material to be fused can descend.

Any change in the temperature of the combustion-zone will change the position of the fusion-zones, an increase in temperature causing them to move upward, and a decrease causing them to move downward. Increasing the burden would also somewhat lower the fusion-zone.

It is probable that much of the trouble from hangs, slips, and scaffolds can be traced to these shiftings of the fusion-zones. It would seem to be of the greatest advantage to be able to maintain the fusion-zones as nearly stationary as possible, and in order to bring this about, with a given burden, it is necessary to maintain a uniform temperature of combustion.

Much has been said about the utilization of the heat-units delivered to the furnace. But much more important is the consideration of the temperatures produced by these heat-units. Keep the temperatures right, and the heat-units have a tendency to take care of themselves. It would matter little how much fuel were burned, or how many heat-units furnished to the blast-furnace, if the hearth-temperature were not maintained at a point above the melting-point of the slag produced. Keep the temperature above this point and the furnace will run. The higher the temperature the more rapidly it will run, provided there is not so much fuel present as to hinder the descent of fresh material.

Combustion-temperature is, then, the important point, the amount of slag and iron melted in a given time depending almost wholly upon the surface exposed and the temperature of the gases passing around the melting particles.

The factors to be considered in determining combustion-tem-

peratures are the reactions of combustion and the temperature of the reacting substances. When these are uniform the temperature of combustion will be uniform; but if, for any cause, there are variations in any factor the temperature will be immediately affected.

If we assume that in the blast-furnace the conditions are maintained as nearly constant as possible, that the burden remains the same, the amounts of slag and pig-iron are the same, and the reactions involved in their formation and fusion are consequently the same, then it will be the reactions of combustion of the solid carbon which affect the hearth-temperature; and it may be expected that the temperature of the hearth will vary directly as the temperature of combustion of the solid carbon, being, of course, somewhat lower, but by an amount which is bound to be almost constant for a given set of conditions.

The factors which control the combustion-temperature of solid carbon in the blast-furnace are the composition of the air used and the temperature to which it has been preheated. The temperature obtained will be the result of the heating of the gaseous products of the combustion by the heat which is available, from the combustion of the carbon, from the sensible heat in the air, and from the sensible heat in the solid carbon at the moment of combustion. (Since the air contains more or less water-vapor, this must be taken into consideration.) It may be assumed that the carbon particles have become heated to the temperature of combustion at the moment of the combustion itself.

In order to investigate the relations between temperature of combustion, blast-temperature, and moisture-content of the air used, I have calculated the combustion-temperatures of solid carbon for the following four conditions:

a. Blast-temperature, 0° C., dry blast, ...	1,675° C.
b. Blast-temperature, 1,000° C., dry blast, ...	2,150° C.
c. Blast-temperature, 0° C., 20 grains moisture per cubic foot, .	1,360° C.
d. Blast-temperature, 1,000° C., 20 grains moisture per cubic foot, .	2,260° C.

Since most American furnace-men prefer to deal with the Fahrenheit scale, I have also calculated the following values:

a. Blast-temperature, 0° F., no moisture, . . .	3,018° F.
b. Blast-temperature, 1,600° F., no moisture, . .	4,353° F.
c. Blast-temperature, 0° F., 20 grains moisture, . .	2,453° F.
d. Blast-temperature, 1,600° F., 20 grains moisture, .	3,885° F.

I do not claim special originality for these calculations. They were based on the work of Prof. Joseph W. Richards, as given in his articles on Metallurgical Calculations, and afterwards in his book on that subject. I have used also the graphic method described by Damour and Queneau, in their book on "Industrial Furnaces." To Mallard and Le Chatelier I am likewise indebted. Specific heats and heats of reaction were taken from Professor Richards's articles in Vol. I. of his book. Since I employed his constants, it is not surprising that my results should closely coincide with his. But I was not aware that he had made calculations of the combustion-temperature of carbon under various conditions until I saw them in his second volume after my own were finished. That part of this paper for which I may claim originality is the diagram showing the relations between the moisture-content of the air, the blast-temperature, and the resulting theoretical temperature of combustion; and the application of this diagram to the problem of the control of combustion-temperatures, and thus, incidentally, to the attainment of uniformity in blast-furnace operations.

Using the values given in Fahrenheit degrees, I constructed a chart (Fig. 1) showing the relations for all moisture-contents up to 20 grains per cubic foot, and for all temperatures up to 1,600° F.

This chart is not strictly correct, since it assumes that all temperatures of combustion will be proportional to the differences from the figures given. As a matter of fact, there is a slight variation, but not enough to make any essential difference in the results. The combustion-temperature lines should show a slight upward curve.

This chart shows that changes in combustion-temperature may be due to either or both of two things. If the blast-temperature be maintained constant, the combustion-temperature will vary with the amount of moisture. Roughly speaking, it may be said that each grain of moisture per cubic foot of air will lower the combustion-temperature about 25° F. If the amount of moisture remains constant, increase of the blast-temperature will raise the combustion-temperature, the amount being, roughly, 85° F. for every 100° F. increase in the blast-temperature. It may also be seen that the decrease in combustion-temperature due to 1 grain of moisture may be compensated by carrying a blast-temperature about 30° higher.

Most furnace-managers make some attempt to regulate the

hearth-temperature by controlling the blast-temperature, but they do this only by observing the operation of the furnace, and making the correction by raising the blast-temperature after it has become evident that the hearth is cooling. If the blast-temperature could be so maintained that the cooling did not take place at all, the practice would be much better.[1]

The problem is solved in the Gayley dry-blast plants by maintaining a uniform moisture-content, thus making it a simple matter to maintain, by means of a very nearly uniform

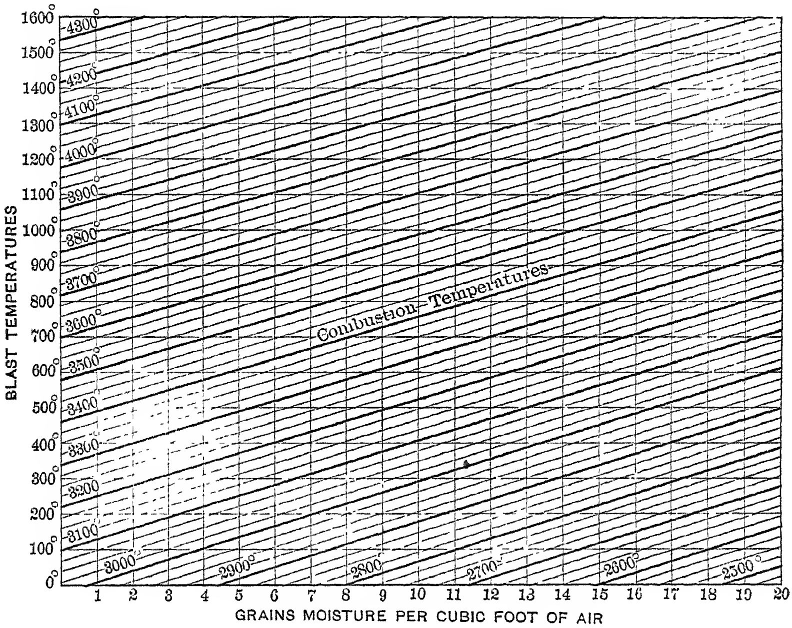

FIG. 1.—CHART SHOWING RELATIONS OF BLAST-TEMPERATURE, MOISTURE-CONTENT, AND COMBUSTION-TEMPERATURE.

blast-temperature, a nearly uniform combustion-temperature, thus securing a practically uniform operation of the furnace.

For the plants that have not as yet installed desiccating-plants for the air, it is probable that much benefit would follow a more careful attempt to control the temperature of the hearth through hot-blast manipulation. This has been suggested before, but has generally been put aside as not valuable enough to compensate for the trouble involved. Yet it would seem to be a simple and not very difficult matter if undertaken by careful men and in the proper spirit.

An experiment in such temperature-control would surely be well worth the time and trouble involved; and for such an experiment the following plan is suggested:

Let a blast-furnace be selected where the burden can be maintained fairly uniform and the fuel-supply fairly regular. Ample stove-capacity should be available, so that there shall be no difficulty in increasing the blast-temperature when needed. Arrangements for the admixture of cold blast should be provided, and the aim should be to maintain the blast at a constant temperature through cold-blast admixture rather than to suffer the variations due to the cooling of the stoves, and the changing from a cold stove to a hot one.

Moisture-determinations should be taken every hour with a sling-psychrometer in the current of air drawn into the blowing-tubs. If more than one engine is running on a furnace, the average of determinations taken at the different engines should be taken; and a certain combustion-temperature should be agreed upon as the proper one to be maintained during the test.

From the moisture-determinations, it will be a simple matter to determine the proper blast-temperature which should be maintained. This can be done by reference to such a chart as is shown in Fig. 1. The stove-tender should then be instructed to maintain this temperature until the next reading is taken.

Such an experiment is surely well worth the trial, and might help to solve the problems due to irregularity of air-composition and combustion-temperatures. If uniformity is the principal reason for the success of the Gayley process, similar advantages should be obtained from the uniformity secured by the control suggested above. The effects, however, of the entire absence of moisture in the blast, and the absence of hydrogen in the combustion-gases, cannot be thus controlled.

In order to illustrate the close relations between combustion-temperature and furnace-operation, the following results, obtained from a recent blast-furnace test, are given:

The test was made in February, 1909, on a 300-ton furnace, covering the operation of the furnace for 72 hr., and was primarily designed to determine the heat-distribution.

The necessary data were obtained by hourly determinations

of the moisture of the blast; hourly analyses of downcomer-gases; continuous record of blast-temperature taken by a re-

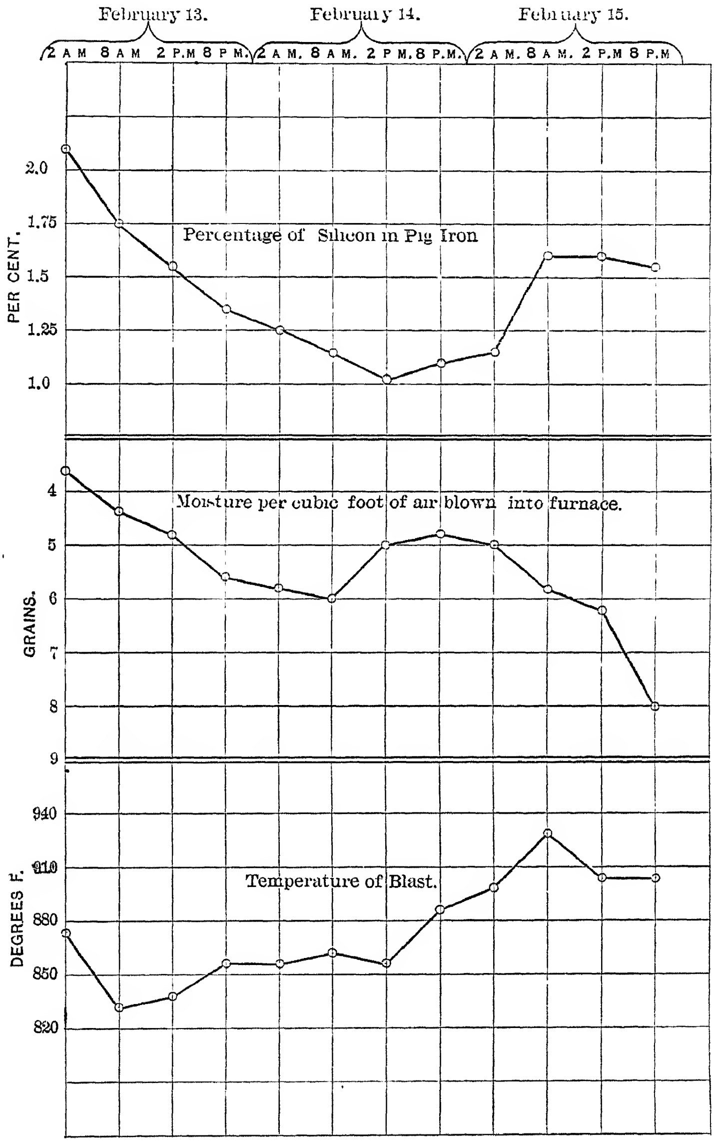

Fig. 2.—Chart of Average Results of Blast-Furnace Operations.

cording-pyrometer; record of weight and analyses of samples of each cast of pig-iron; samples and analyses of each flush of

cinder, accompanied with determination of the temperature by the Fèry radiation pyrometer.

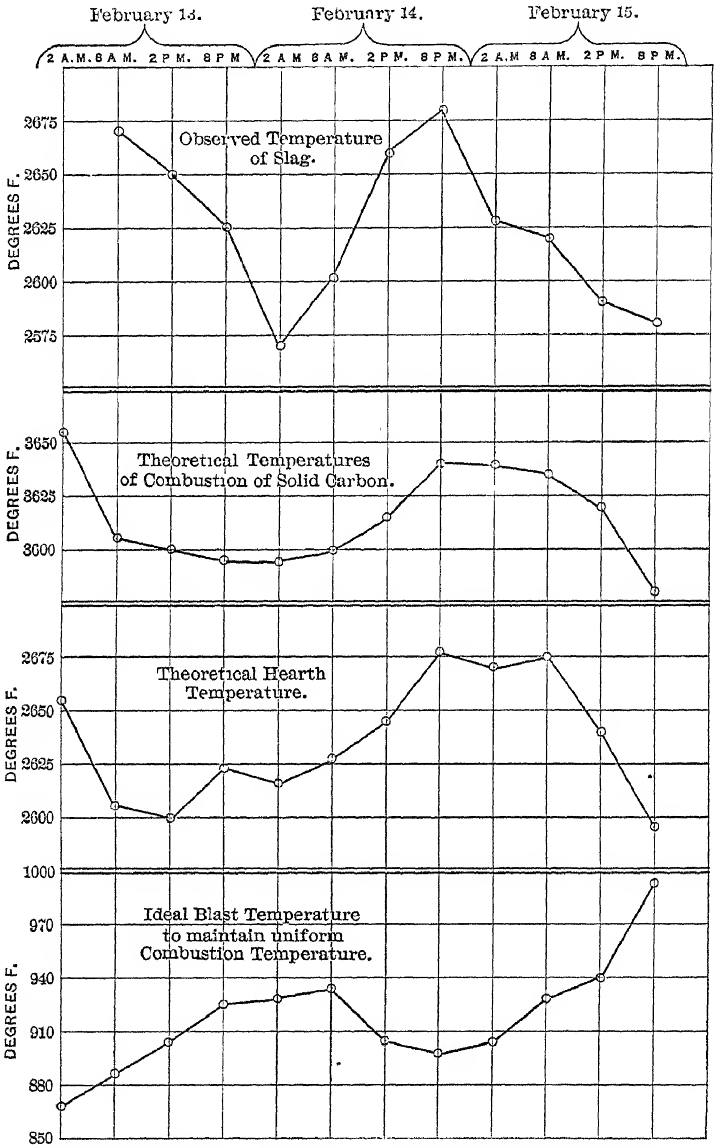

FIG. 3.—CHART OF AVERAGE RESULTS OF BLAST-FURNACE OPERATIONS.

In order to investigate the effects of combustion-temperatures on the working of the furnace, calculations have been made

of the theoretical combustion-temperatures during the test, and the results obtained have been compared with those of the furnace-operation.

Hearth-temperature cannot as yet be judged by any direct means, so we have to rely upon certain indirect methods for determining relative hearth-temperatures.

With a uniform burden and slag-composition, the percentage of silicon in the pig-iron may be taken as a fair indication of the temperature. The higher the silicon the higher the hearth-temperature; hence the percentage of silicon in the iron from cast to cast should follow somewhat closely the theoretical combustion-temperature.

The temperature of the slag flowing from the furnace should also depend upon the combustion-temperature, and should vary, other things being uniform, with it. It will be interesting, then, to compare the results found with these assumptions.

The readings and calculations have been averaged for the periods between the casts, and the results have been plotted as shown in Figs. 2 and 3.

In these diagrams the curves show: the percentage of silicon in each cast; the moisture in the blast, in grains per cubic foot; the temperature of the blast; the observed temperature of the slag; the theoretical combustion-temperature; the theoretical hearth-temperature, corrected for the basicity of the slag; and the ideal blast-temperature for maintaining a uniform temperature of combustion of 3,650° F.

In reference to these curves, it may be explained that an attempt has been made to plot them on a uniform scale, so that the results will be easily comparable.

A slight change in burden was made at the end of the first 24 hr., lowering the amount of iron but increasing the amount of slag.

The slag-temperatures were taken as carefully as possible; but in some cases it was difficult to secure accurate readings, fumes arising in clouds from the surface of the slag so as to obstruct some of the heat-rays. It is thought, however, that the averages shown are very close to the proper values.

The line representing theoretical hearth-temperatures is drawn empirically only. It assumes that the hearth-tempera-

ture is about that of the slag, or about 1,000° lower than the combustion-temperature of the solid carbon.

Allowance was also made for the differences in fusion-temperatures of the slags, it being found that a difference of about 70° F. existed between the more basic and the less basic. The mean of these was taken as a base-line for corrections, and the degrees of the fusing-temperature above or below this mean were added to or subtracted from the combustion-temperature after subtracting 1,000°.

It will be noted that at the beginning of the test the percentage of silicon in the pig is fairly high; that it rapidly decreases for about a day and a half, when it rises to a maximum at the end of two days and a half, then falling somewhat at the end of the test.

A remarkable similarity will be observed among all of the curves in that their general shape is the same, and their maxima and minima correspond very well with each other and with the silicon-curve.

The curve for observed slag-temperature follows almost exactly the corrected hearth-temperature curve, as indeed it should do.

The effect of moisture on the slag-temperature is well illustrated; its effect on hearth-temperature evidently being somewhat greater than the calculations show. The amount of moisture varied greatly during the test, running from 3.7 grains per cubic foot at the beginning to a little more than 8 grains at the end. The bad effects of this excessive moisture were, however, largely counteracted by the gradually increasing blast-temperature.

The conclusions to be drawn from the test, so far as hearth-temperatures are concerned, seem to be about as follows:

The effects of lowering the theoretical temperature of combustion are to give a cooler slag and an iron lower in silicon.

The lowering of the silicon follows very closely the fall of the theoretical temperature of combustion, but the raising seems to lag somewhat behind the increase of the combustion-temperature.

The effects of moisture in the blast may be compensated by increasing the blast-temperature, so far as it concerns hearth-

temperature, the temperature of the slag, and the composition of the pig-iron.

The temperature of the slag follows very closely the calculated combustion-temperature, being, of course, much lower.

In connection with the regulation of the hearth-temperature, the last curve is given as an illustration of what the blast-temperature should have been, so as to give a uniform combustion-temperature of 3,650° F., this being about the temperature at the beginning of the test. The temperatures used for this curve are as follows:

	Moisture. Grains.	Blast-Temperature. Degrees F.		Moisture. Grains.	Blast-Temperature. Degrees F.
1, . . .	3.7	868	7, . . .	5.0	908
2, . . .	4.4	886	8, . . .	4.8	900
3, . . .	4.9	904	9, . . .	5.0	908
4, . . .	5.7	925	10, . . .	5.8	928
5, . . .	5.8	928	11, . . .	6.2	940
6, . . .	6.0	935	12, . . .	8.0	992

If we should assume that the amount of moisture had been held uniform at 2 grains per cubic foot, the same result could have been obtained by maintaining a blast-temperature of 820° F.

Introduction of the Thomas Basic Steel Process in the United States.

BY GEORGE W. MAYNARD, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910.)

AT the Pittsburg meeting of the Institute, May, 1879, I made the first announcement in America of the results obtained by Sidney Gilchrist Thomas and Percy C. Gilchrist, in their efforts to eliminate phosphorus in the manufacture of steel in the Bessemer converter and the open-hearth furnace.[1]

The first published statement in an American newspaper was a reprint of the first Thomas-Gilchrist paper,[2] which I sent to J. C. Bayles, at that time the editor. At the suggestion of many friends who were present at the Pittsburg meeting, and others who are cognizant of the beginnings of the process in the United States, and for the information of the younger generation of the iron and steel fraternity, I have thought it eminently proper that the history of the process in detail should be given where I made the first announcement and where the process has had its largest development.

During my residence in England, from April, 1873, to February, 1879, and as a member of the Iron and Steel Institute since 1874, I was present at many of its meetings during those years. At nearly every meeting the burden of discussion was the question of the use of pig-iron containing phosphorus in the manufacture of steel by the Bessemer or the open-hearth process.

In my biographical notice of Thomas,[3] read at the New York meeting of the Institute, February, 1885, I show that the working-out of the process was not a hap-hazard or accidental inspiration, but the culmination of many years of investigation of experiments and theories which had been carried on and advanced by chemists and metallurgists at home and abroad.

[1] *Trans.*, viii., 5 (1879–80).

[2] *The Iron Age*, vol. 22, No. 17, p. 3 (Oct. 24, 1878).

[3] *Trans.*, xiii., 785 to 791.

The foundation of Thomas's great achievement was his chemical knowledge, he having passed an examination in inorganic chemistry, "first class advanced," at the Royal School of Mines, as the outcome of attending night-lectures and working in a little laboratory at his cottage in Battersea, where it was my good fortune to spend many evenings with him. He supplemented his chemical work by visits to English, Belgian, and German iron- and steel-works when he could get away for short periods from his exacting duties in the Thames Police Court.

There are numerous papers and discussions by the most eminent metallurgists and chemists on the use of pig-iron containing phosphorus in the manufacture of steel by the Bessemer or the open-hearth process.[4] The theories advanced and the practical tests by Bell, Siemens, Snelus, Crampton, Howson, Riley, Stead, and other investigators in the same field make most interesting reading in view of what has been accomplished in these late years.

A preliminary puddling or "washing" of the phosphorus-bearing pig-iron, whereby the phosphorus was reduced by the addition of an iron oxide or iron-ore in the "washing" process, and then teemed into the Bessemer converter or the open-hearth furnace for the final conversion, was the basis of the many suggestions and experiments; various forms of mechanical puddling-furnaces for reducing costs and accelerating the preliminary treatment being the salient differences between the competing methods. As preliminary to the solving of the problem of phosphorus-elimination, the following references are worthy of reproduction. At the London meeting of the Iron and Steel Institute in 1877, I. Lowthian Bell read a paper, On the Separation of Carbon, Silicon, Sulphur, and Phosphorus in the Refining and Puddling Furnace and in the Bessemer Converter,[5] in which he states:

> "Exposed to the intensely deoxidising agency of the blast furnace, portions of the silica, and probably the greater portion of the sulphur compounds lose their oxygen and are taken up wholly or in part by the reduced iron. Practically, however, it may be assumed that all the phosphoric acid in the minerals is decomposed during the smelting process, and in consequence all its phosphorus is to be found in the pig metal."

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[4] *Journal of the Iron and Steel Institute*, vols. xi. to xv. (1877 to 1879).

[5] *Idem*, vol. xi., pp. 108 to 125 (No. I., 1877).

In a series of experiments he shows the extent to which the phosphorus in Cleveland pig-iron was reduced in the puddling-furnace. The results obtained in further tests are given in the paper with the same title[6] read at the Newcastle meeting in September, 1877:

". . . . I quoted the experience derived from the Bessemer converter to prove that oxygen in its free state, or as oxide of iron, was almost entirely inert as regards phosphorus at the intense temperature which accompanies the Bessemer process. The blast was continued until a large quantity of the original metal was oxidized, and upon the occasion of a subsequent experiment, powdered hematite ore was blown into the iron without producing any very marked effect.

"When, however, melted pig iron was exposed to the action of fluid oxide of iron at lower temperatures, phosphorus was rapidly removed.

"In this operation the phosphorus is, no doubt, converted into phosphoric acid by depriving a portion of the oxide of iron of its oxygen, and the acid so formed immediately combines with a second equivalent of oxide of iron; or, if the metal is in a higher state of oxidation than protoxide, the phosphorus seizes a portion of the oxygen, and the resulting phosphoric acid combines with the newly formed lower or prot-oxide. . . . .

"The facts as now stated justify the assertion that the order of affinity between the metal and phosphorus, by differences of heat alone, is inverted in the manner referred to. . . . . This was ascertained by a variety of modes of procedure. In one instance a specimen of Cleveland iron had been so well freed from its phosphorus that only .045 per cent. of this substance remained. A portion of this purified metal was subsequently exposed to a strong heat, about equal to that of a puddling furnace, in contact with the cinder used in its purification. At the end of 15 minutes the phosphorus had risen to .153 per cent., and in three hours the iron contained .365 per cent. of this element."

Mr. Bell's paper was discussed by Professor Williamson, Edward Riley, Edward Williams, Baker of Sheffield, Snelus, and Stead,[7] and it is a very interesting fact that no one of these eminent authorities suggested lime as a base for taking up and holding the phosphoric acid.

This is all the more remarkable in view of Professor Williamson's statement:[8]

". . . . he ought to remind the meeting that silica at high temperatures expels phosphoric acid from its salts. . . . . If they had got a salt—he meant a compound of phosphoric acid, say, with an oxide of iron—and if they heated that in contact with a reducing agent and with silica, they could turn out the phosphoric acid, and get it reduced to phosphorus in proportion as they turned it out; . . . ."

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[6] *Journal of the Iron and Steel Institute,* vol. xi., pp. 322 to 344 (No. II., 1877).

[7] *Idem,* vol. xi., pp. 359 to 395 (No. II., 1877).

[8] *Idem,* vol. xi., p. 360 (No. II., 1877).

The next meeting of the Institute was held in London, in March, 1878. At that meeting Mr. Bell read a paper, On the Separation of Phosphorus from Pig Iron.[9] Professor Williamson again took part in the discussion, in which he suggested "to diminish the amount of oxide of iron that was used by replacing that part of it which had not to give up oxygen, but merely to act as a base, either by lime or some other base of comparatively little value." In the continuation of the discussion, George J. Snelus said:

"He considered of the utmost consequence one point mentioned by Professor Williamson, who had hit upon a nail which he himself had for a considerable time been trying to drive home. Six years ago he took out a patent for using lime for the lining of steel melting and other furnaces—a patent which was still valid. He had hitherto refrained from saying anything about it at these meetings. Knowing the value of the essence of the thing, he had been trying for the last six years to devise some mechanical means for reducing it to practice. . . . . He had himself lined a Bessemer converter with limestone, and the result . . . . from 3 or 4 tons of iron so treated perfectly corroborated what he had ascertained in dealing with 1½ cwt. of iron—viz., that the phosphorus was almost entirely eliminated under those circumstances. There were, however, he thought, practical difficulties in applying the lime, ..."

The date of the patent to which Mr. Snelus referred is Mar. 25, 1872, and it is entitled, An Improved Lining for Cupola Furnaces, also Applicable to the Formation of the Beds of Reverberatory Furnaces. In the patent there is not even a remote reference to the removal of phosphorus.

Holley, in a report to the Bessemer Steel Co., Ltd., gives the following explanation of Snelus's silence as to the elimination of phosphorus, after quoting his claims:

"Had not Snelus been in the employ of a company specially interested in keeping the Bessemer trade on the west coast of England he would now occupy the place of Thomas. He worked out the problem as a chemist and knew perfectly well the full value of his invention, but being afraid of antagonizing his employers, the West Cumberland Company, he dared not fully explain the purport of his discovery."

In making this comment Holley evidently lost sight of the "after-blow," which was the salvation of the basic Bessemer process, of which there is no evidence that it had occurred to Snelus, and Riley's substitution of tar for making the burnt dolomite plastic.

[9] *Journal of the Iron and Steel Institute*, vol. xii., pp. 17 to 34 (No. I., 1878).

These two discoveries were as vital to the basic process as Mushet's spiegeleisen and ferro-manganese to the Bessemer process.

In the final outcome Thomas generously recognized Snelus's early experiments.

Among those who took part in the discussion at the London meeting, March, 1878, was a young man, a visitor, who said that he had succeeded in effecting the almost complete removal of phosphorus in the Bessemer process. Experiments had been carried out at Blaenavon, with the co-operation of E. P. Martin, on quantities varying from 6 lb. to 10 cwt., and some hundred analyses made by Mr. Gilchrist (who had the conduct of the experiment from the first), showing the removal of from 20 to 99.9 per cent. of the phosphorus in the converter. He believed the practical difficulties in the way had been overcome, and that Cleveland pig-iron might be made into good steel without any intermediate process; he hoped on a future occasion to lay full details before the Institute. The "young man" was Sidney Gilchrist Thomas.

Mr. Jeans, who was present, and subsequently became the Secretary of the Iron and Steel Institute, in his work[10] says: "The meeting did not laugh at this youthful *Eureka*, nor did it congratulate the young man on his achievement. Much less did it inquire about his method of elimination; it simply took no notice of his undemonstrative announcement."

On Saturday, Sept. 14, 1878, A. L. Holley and I had planned to leave London to attend the Paris meeting of the Iron and Steel Institute. As we were about starting for the Charing Cross station, I was summoned by wire to the north of England on some engineering work. We parted with the expectation of meeting in Paris within a day or two. My work prevented my going. Within a week I received a letter from Mr. Holley inclosing a paper which had been submitted to the Council of the Institute for presentation at the Paris meeting. The title of the paper was, On the Elimination of Phosphorus in the Bessemer Converter, by Sidney Gilchrist Thomas, F.C.S., and Percy C. Gilchrist, Associate Royal School of Mines, F.C.S., published in English and French.

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[10] *Creators of the Age of Steel*, p. 304 (1884).

Mr. Holley, in his letter, wrote: "This looks all right, and if upon examination you find it to be so, you had better get control of the process for the United States."

The simplicity of the chemistry of the process and its great possibilities immediately appealed to me, so without delay I telegraphed to Mr. Thomas through Mr. Deby, the Foreign Secretary of the Institute, to call on me on his return to London with a view to taking up the process for the United States. To quote from R. W. Burnie's memoir of S. G. Thomas: "The paper was not read at the Paris meeting, although it had originally been placed near the top of the list, but belief in the alleged discovery of an unknown youth had not much spread since March, and the paper was removed to the end, and then left by the authorities unread for 'lack of time.'"

Windsor Richards, manager of the works of Bolckow, Vaughan & Co., at Middlesbrough, was at the Paris meeting; and in his Presidential address to the Cleveland Institution of Engineers, he said:

"Messrs. Thomas and Gilchrist prepared a paper giving very fully the results of their experiments with analyses. It was intended to be read at the Paris meeting in 1878, but so little importance was attached to it, and so little was it believed in, that the paper was scarcely noticed, and was left unread. Mr. Thomas first drew my particular attention to the subject at Creusot, and we had a meeting a few days later at Paris, when I resolved to take the matter up, provided I received the consent of my directors. That consent was given, and on Oct. 2, 1878, accompanied by Mr. Stead of Middlesbrough, I went with Mr. Thomas to Blaenavon. On arriving there Mr. Gilchrist and Mr. Martin showed us three casts from a miniature cupola, and I saw enough to convince me that iron could be dephosphorized at a high temperature. I also visited the Dowlais works, where Mr. Menelaus informed me that the experiments in the large converters had failed owing to the lining being washed out."

The outcome of Mr. Richards's investigation was the erection of a pair of 30-cwt. converters at the works of Bolckow, Vaughan & Co. at Middlesbrough. At the beginning they were confronted with the difficulty of making basic brick for the converters. Their difficulties were finally overcome, so that on April 4, 1879, they were able to show the iron manufacturers at Middlesbrough an absolutely successful operation.

On Mr. Thomas's return to London in September, 1878, he called on me at my request with a view to a personal investi-

gation of the process and its adoption in the United States. At that time I was consulting engineer for the Standard Iron & Steel Co., a Manchester corporation, with works at Gorton, a suburb of Manchester.

As a substitute for a two-converter Bessemer plant the company had been induced to adopt a process which was to produce steel from any grade of pig-iron. The practical demonstration promptly resulted in the abandonment of the process, leaving, however, a "receiver," an oblong ganister-lined box, with tuyeres on both sides, somewhat on the lines of the Clapp-Griffiths plant.

I suggested to Mr. Thomas that a test of his process be made in the receiver, which he thought would be quite feasible and much less costly than to line a Bessemer converter.

On consultation with the directors of the Standard Co., they authorized me to go ahead, and voted the necessary funds for carrying on the investigation.

In order to have the test as severe as possible, I concluded to make the demonstration at Middlesbrough.

The proprietors of the Acklam Works very kindly permitted me to set up the receiver at the lower end of the pig-bed; the blast pressure of from 2.5 to 3.5 lb. was furnished by the blast-furnace blowing-engine. Messrs. Thomas and Gilchrist and Mr. Stead, the eminent chemist of Middlesbrough, were present and some of the officials of the Acklam Works.

The basic bricks (dolomite) were burnt in an ordinary brick-kiln, and, as it later transpired, at too low a temperature.

The first heat was made Dec. 18, 1878, and the second on Dec. 20. In both cases the charge was 5 tons of No. 3 iron containing 1.63 per cent. of phosphorus. In the first charge the resultant metal contained 0.45 per cent. of phosphorus and no silicon; the cinder, 7.05 per cent. of phosphoric acid with 31 per cent. of silica. In the second charge, with the same grade of iron, the product contained 0.28 per cent. of phosphorus and a trace of silicon. The samples were taken by Mr. Stead, who also made the analyses.

The product was a bad-looking mess, as much of the basic lining floated out of the box along with the steel. I believe these blows were the first after the South Wales experiments.

Shortly after, a plant was erected at Eston, and to Windsor

Richards is due the credit of having made the first commercial success, and for the facilities which he afforded English and foreign steel manufacturers for studying the process.

The analyses given in Table I. show the results of the early experiments, as carried out by Mr. Richards, which confirmed triumphantly Mr. Thomas's theory and predictions.

TABLE I.—*Results of Experiments at Bolckow, Vaughan & Co.'s Eston Works.*

Original Pig-Iron.	Metal after Blowing for				
	6 Minutes.	12 Minutes.	14 5 Minutes. End of Blow.	16 5 Minutes	16 Minutes, 35 Seconds. End After-Blow.
Per cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Carbon 3.57	3.40	0.88	0.07	trace.	trace.
Silicon.................1.70	0.28	0.01	trace.	nil.	nil.
Phosphorus 1.57	1.63	1.42	1.22	0.14	0.08
Manganese............0.71	0.56	0.27	0.12	0.10	trace.
Sulphur... ....0.06	0.06	0.05	0.05	0.05	0.05

The Acklam experience confirmed me in my wish to take up the process for America. Then followed almost daily conferences with Mr. Thomas on patents and working-details up to the time of leaving for New York in March, 1879.

In the manufacture of basic brick or furnace-linings the serious drawback was the slacking of the lime when water was used. This difficulty was overcome by the knowledge and ingenuity of Edward Riley, the eminent metallurgical chemist, by substituting gas-tar or petroleum for the purpose of making the hard-burnt dolomite plastic before molding.

I was with Mr. Riley during many of his experiments. Patents were finally granted him on Nov. 25, 1878. I was appointed by Mr. Thomas his sole agent for the United States for the granting of licenses or sale of the patents, and for the securing of patents which had not yet gone to issue. Subsequently I was empowered by Mr. Riley to act for him. It is extremely gratifying to inform you that one of our members, Horace W. Lash, formerly of Pittsburg and now of Cleveland, was the first man to test the basic process in an open-hearth furnace. Mr. Lash has kindly furnished me with a statement of his work in Belgium in the following letter:

"CLEVELAND, OHIO, December 16, 1909.

"GEO. W. MAYNARD, ESQ., New York City, N. Y.

"'My dear Mr. Maynard: In reply to your letter of December 9th in reference to my work in Belgium in connection with the Basic Process, would say I met Mr. Thomas at the Autumn meeting of the Iron and Steel Institute, which was held in Paris in October, 1878.

"Up to this time Mr. Thomas had done but little, if anything, with his lining in the open-hearth furnace, his experiments being confined largely to the Bessemer converter in the old plant at Dowlais, South Wales. I advised Mr. Thomas that in my opinion the Ponsard furnace, with which he had been making a number of experiments at Thy-le-Chateau, Belgium, would be a good furnace in which to try his process, as it would be possible to line the bottom or revolving part of the furnace with his lining, allowing the roof and ports to remain the usual construction of silica brick. In this way there would be a natural dividing line between the two materials, which division or separation appeared to be a serious problem at that time. Mr. Thomas was highly impressed with the possibilities of working his process in this type of furnace. In consideration of this we proceeded to Thy-le-Chateau, where we spent several days in looking up suitable material for lining. This we found in a highly magnesian limestone near-by the works. We had a quantity of this material quarried and delivered at the experimental plant, where it was ground into a fine condition and mixed with about 3 per cent. of sodium silicate. With this mixture we made a rammed lining of from 16 to 18 in. thick. Our first trial heats consisted of about 2 tons of old steel rails and 4 tons of local pig-iron, which latter carried about 2 per cent. of phosphorus. The first heat analyzed 0.14 and the second heat 0.30 per cent. of phosphorus. The high phosphorus in the second heat no doubt was largely due to some of the silica bricks, which toppled from the edge of the ports into the bath.

"Mr. Thomas was very highly pleased with the results, and after running several more heats we had the furnace relined and made some further trials, of which I have mislaid the records.

"These trials were all made during the month of November and the early part of December, 1878. Shortly after this I left for London, where I met you with Mr. Thomas, at which time you will no doubt remember Mr. Thomas writing the letter inclosed herewith, dated Dec. 14, 1878, agreeing to pay me certain royalties, after deducting your regular commission, providing I succeeded in introducing this process with certain open-hearth furnace people in the United States.

"I reached this country the early part of 1879, after which I had considerable correspondence with Mr. Thomas, but owing to the disinclination of the American steel manufacturers to take up the basic process in connection with the open-hearth furnaces, I allowed the matter to drop, and thereafter took but little interest in the introduction of the basic steel process in this country or elsewhere.

"Trusting this information may be of some service to you, believe me,

"Yours very truly,

(Signed) "HORACE W. LASH."

After my return to New York in March, 1879, until the following May, much of my time was spent at the Patent Office in Washington, in an effort to educate the Metallurgical Department. Many of the interferences entertained by the Department had no direct bearing on the claims set forth in

Thomas's patent applications. Finally the five foundation claims of Mr. Thomas were patented. During this period I received many applications for licenses on the patents going to issue. The first license was granted to Shoenberger & Co., of Pittsburg, with the intention of using a basic lining for their open-hearth furnaces. I am not aware that they adopted the process during the life of the patents.

The first open-hearth tests were made at the works of the Otis Steel Co., in Cleveland, by S. T. Wellman, who was then the manager of the works. Mr Wellman has been so good as to furnish me with the following statements:

"I duly received yours of Dec. 19, 1909. While traveling in Europe in 1885, I saw the first basic open-hearth steel made. I was so interested in it and so impressed with the high-class product which they were able to make out of the very impure iron, that I purchased from Carl Später at Coblenz a small cargo of 800 tons of raw Styrian magnesite. We did not do anything with it until the following year, when I put in the first bottom in one of our 15-ton open-hearth furnaces. I did not take out the old silica lining, but simply burned it down 6 or 8 in. below the usual line. I then rammed up the bottom with magnesite and tar, the magnesite having been previously dead-burned in one of our open-hearth furnaces. This bottom did not do very well, as the burning out of the tar made it so porous that the steel was constantly breaking through and giving us trouble. We ran the furnace several weeks with varying success. We did not use a very high phosphorus pig-iron, most of it being the ordinary Bessemer pig, containing from 0.10 to 0.15 per cent. of phosphorus. This comprised about half of the charge, the balance being ordinary cheap miscellaneous steel scrap.

"Nearly all of our product was first-class in every respect, being fully equal and some of it superior to the steel made from Northern New York charcoal-blooms, which was the material which we generally used in our high-class steel. The product was very satisfactory, but the output of the furnaces was considerably less in quantity than what we could make by using the ordinary acid steel process used in high-class material. On that account we finally gave up the use of the process for the time being, as we were behind our orders and it was very important that our works turn out the largest amount of steel possible. It is interesting to me to note that to-day practically all the basic open-hearth furnaces in the United States use magnesite from the identical mines from which I obtained this small cargo. Practically the only difference between our practice, in these first experiments, and the standard practice of to-day, is that, in making the bottom, the finely-ground burnt magnesite is burnt into the bottom in thin layers, treating it exactly as the same quantity of silica sand in making an acid steel bottom. The practice in charging furnaces to-day is to charge as much of the material as possible in the initial charge, the bulk of the limestone necesssary for the dephosphorization being put in with the charge. Our practice in those days was to add the limestone after the material was all charged. We also tried to remove the slag, thinking it was necessary in order to get good dephosphorization.

"This is about all there is to say in regard to our first experiments.

"Yours very truly,

(Signed) "S. T. WELLMAN."

In the latter part of September, 1879, Mr. Carnegie arrived in New York from England, and informed me that he had called on Thomas in London and had contracted with him to purchase the basic patents for the United States for a certain amount in cash, and that it was his intention to sell the patents to the Bessemer Co., Ltd., which company was composed of the original 11 Bessemer plants. The outcome of many conferences with Mr. Carnegie was my offer of the patents to the Bessemer Co., which involved many meetings in Philadelphia. The offer embraced the Snelus patent, controlled by Abram S. Hewitt and Edward Cooper, together with the Riley and Thomas patents. I granted an option of purchase for a specific period, during which time Mr. Holley, the consulting engineer of the Bessemer Co., went to Europe to visit the plants which had already adopted the process. The option period was subsequently extended for two months on the advance payment of a portion of the purchase-price.

Thomas arrived in New York the latter part of March. Mr. Hewitt had been selected by all the parties in interest to distribute the final payments. The dinner given to Mr. Thomas at Delmonico's and presided over by Mr. Hewitt was a memorable one, the principal steel-works of the country being represented by their officials. Thomas's letter of April 2, 1881, addressed to his mother, is worthy of reproduction.

"Saturday, April 2.

"The dinner is happily past and I actually enjoyed it, partly. It was dreadful sitting for three hours and being be-praised, but the speakers were really clever and witty in the extreme—alternating between flights of real eloquence and the most fanciful word-fun and wildest jokes. The actual dinner was, of course, superb, costing about £200. I got through my speech fairly I think. I had brought over a first-class one, but couldn't think of a bit so started on quite another line. (Signed) "SIDNEY."

The guests at the dinner were:

Sidney Gilchrist Thomas, Charles F. Chandler, Robert L. Fowler, Chester Griswold, George A. Crocker, Herman Kobbe, Frank S. Witherbee, Stephen W. Baldwin, A. W. Humphreys, R. W. Raymond, D. Van Nostrand, Edward Cooper, James A. Burden, Abram S. Hewitt, Henry Morton, Eckley B. Coxe, Thomas M. Drown, John Bogart, James C. Bayles, ——— Thomson, Gen. John Newton, G. H. Frost, Robert H. Thurston,

B. G. Clarke, J. M. Toucey, Charles Macdonald, Lenox Smith, Cyrus Butler, L. G. Laureau, L. B. Moore, M. N. Forney, Robert W. Hunt, D. S. Hines, E. D. Leavitt, Octave Chanute, Alexander L. Holley, William R. Bunker, George W. Maynard, ——— Brendlinger, Andrew Carnegie, W. B. Crocker, William G. Hamilton, John Fritz.

Not until the spring of 1883 was the manufacture of basic Bessemer steel undertaken. This was at Steelton, at the works of the Pennsylvania Steel Co. I am indebted to John C. Jay, Jr., of this company, and John S. Kennedy, formerly superintendent of the blast-furnace department of this company, for the following data:

"I was superintendent of the blast-furnace department of the Pennsylvania Steel Co. at Steelton for a number of years. On the completion of the new Bessemer plant, the old one containing two 5-ton converters was available for experiments and basic Bessemer steel was there made in 1883. The pig-iron for this basic steel was nearly all made in our No. 5 furnace, a small furnace making about 50 tons daily. The balance was Montour pig-iron, made probably from puddle-cinder, analyzing from 3.50 to 3.799 per cent. of phosphorus, but about 0.06 per cent. of manganese. Our basic pig-iron was not so high in phosphorus but quite high in manganese, as the following analyses will show:

Silicon. Per Cent.	Phosphorus. Per Cent.	Manganese. Per Cent.	Sulphur. Per Cent.
0.48	2.094	2.740	0.050
1.15	1.817	4.080	0.030
1.32	1.80	4.08	0.03
1.28	1.754	2.580	0.035

"We then increased the percentage of phosphorus in the pig-iron and reduced the manganese, using small quantities of converter scrap and slag for the above purposes. The pig-iron then ran:

Silicon. Per Cent.	Phosphorus. Per Cent.	Manganese. Per Cent.	Sulphur. Per Cent.
0.62	3.833	1.800	0.030
0.87	3.704	1.770	0.050
0.35	3.940	2.380	0.060
0.53	3.734	2.440	0.050
0.48	1.373	3.811	0.030

"This was about the iron we intended to make, but variations in the phosphorus and managanese in the mill-cinder and ore used made the iron very irregular in these elements.

"No. 5 furnace was burned down Nov. 21, 1883. The steel made was of excellent quality and very soft. The process proved to be expensive and was abandoned, (Signed) "JOHN S. KENNEDY."

"The first basic Bessemer steel, made at Steelton, was made in our No. 1 Bessemer converter May 7, 1883. The pig-iron for this was made at the old Docklow furnace, which stood where the steel-foundry is at present. We also used some Lochiel pig-iron made when this furnace was owned by John C. Denny & Co.

"This iron contained 3.5 per cent. of phosphorus, less than 1 per cent. of silicon, and about 1 per cent. of manganese.

"The length of blow was from 20 to 30 min.; in most cases this steel took from 5 to 8 min. over-blow before the phosphorus was eliminated.

"The lining was made from a magnesian limestone containing about 45 per cent. of magnesia. This was calcined in a kiln, then ground to the size of a pea, mixed with anhydrous tar so as to hold together, rammed in the vessel, making a complete basic lining. The analysis of the steel made was, C, 0.1; Mn, 0.50; P, 0.04; and S, 0.050. This analysis is as near as we can remember.

"The steel made with this process was very wild in molds, due to too much overblow getting rid of the phosphorus. Trusting that this information will be of help to you, we remain, (Signed) "JOHN JAY, JR."

The second basic Bessemer plant was erected at the works of the Pottstown Iron Co. For the following data I am indebted to Joseph Hartshorne, at that time the manager of the works:

"Started building June, 1885. Started blowing July 1, 1886. Stood idle from August, 1888, to August, 1890. Shut down August, 1893. Double turn from Sept. 1, 1892, to end. I cannot give you the total tonnage, as the records are not in my possession. It may have been between 300,000 and 500,000 tons.

"The shutting-down of the works followed the failure of the company in February, 1893. The principal factor was the cost of pig-iron. When we started we had $2.50 in our favor between the price of the Bessemer and our own pig-iron; when we stopped Bessemer sold for 60 cents a ton less than we could make our pig-iron for; and our process cost about $1 more than the Bessemer. The inference is obvious, due in a great measure to Mesabi.

(Signed) "JOSEPH HARTSHORNE."

A full description of the plant was given by Mr. Hartshorne in his admirable and instructive paper read at the October, 1892, meeting of the Institute at Reading. According to that paper the phosphorus in the pig was from 2.50 to 3 per cent., and in the finished steel from 0.02 to 0.05 per cent.

The third basic Bessemer plant was erected at the works of the Troy Steel Co., June, 1897. For the following data I am indebted to E. D. Arnold, at that time the general manager of the works.

The pig-iron was made almost exclusively from "Old Bed" ore, Lake Champlain, of which the average phosphoric acid content was 1.94 per cent. and the phosphorus in the pig-iron ranged from 2.18 to 2.62 per cent.

A typical day's work, 24 hours, was 32 heats, Dec. 20, 1897.

Average length for a blow, 10 min.

Average length after blow, 3 min.

Average resultant steel, C, 0.0586; P, 0.054 per cent.

The campaign lasted about six months and the total output of ingots was 60,000 tons. The Troy plant was discontinued for financial reasons.

Last year's statistics almost confirm Holley's prophecy that the time would come when "the open-hearth would attend the funeral of the Bessemer," the output being, open-hearth, 14,493,936, and Bessemer, 9,330,783 tons.

The four inventions which have revolutionized the iron and steel manufacture of the world are to be credited to four nationalities.

Bessemer, although born in England, was the son of a Frenchman; the Siemens brothers, citizens of England, but born in Hanover; Thomas, a full-blooded Englishman; and last, but not least, James Gayley, an American—all within a period of 50 years.

Postscript.

Since the publication of the pamphlet edition of my paper I have received a letter from Edward P. Martin from Abergavenny, dated Sept. 7, 1910, from which I quote:

"There are some things that require amplification and explanation, and I do not quite agree with what Holley said of Snelus, but there are other points that would take a great deal of time and work to put clearly before the public the troubles that Thomas went through before he and Gilchrist and others made the process a success."

After the receipt of Mr. Martin's letter, notice of his sudden death was received.

I now realize that I have not paid him the tribute which is his due in extending to Thomas and Gilchrist financial aid and facilities for carrying on experiments at Blaenavon. Mr. Martin was the manager of the Blaenavon Works, and Percy C. Gilchrist, Thomas's cousin, was the works chemist. As Thomas had no facilities in London, he arranged with Gilchrist to help

him out in a practical way. I now quote from R. W. Burnie's *Memoir of Sidney Gilchrist Thomas:*

"At the end of 1877 and the beginning of 1878 the results of the experiments which had been continued for something like nine months with constant energy and zeal had proved thoroughly satisfactory. After trials in crucibles, a miniature converter had been obtained, which, although it only held 8 lb., instead of 8 tons, sufficed for experimental purposes. Northampton pig-iron had been partially dephosphorised by lining the converter with bricks of lime and with silicate of soda. For some time, however, from some defect in the apparatus, the experimentalists were not able to get a fluid cast, so as to finish the operation. Later in the year complete success was achieved, and they obtained a number of casts of 8 lb. each, which, upon analysis, were found to be excellent steel. Just at this time Mr. Martin said to Gilchrist, 'I know you young men have some secret work on hand. I think it would be well if you put confidence in me.' Confidence *was* put in him, and Gilchrist's analyses were submitted to him. Mr. Martin was so much struck with the basic theory and the proof afforded of its truth that he at once contributed for further experiments on a larger scale. He also undertook personally to purchase a share in the patent."

Chemical Laboratories in Iron- and Steel-Works.

BY GEORGE W. MAYNARD, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910.)

In the biographical notice of Thomas F. Witherbee, published in *Bulletin No.* 32, August, 1909 (p. xxv), it is said that "he is believed to have been the first manager in America to use the chemical laboratory for the purpose of controlling the regular running of the blast-furnace."

Since the year is not given, I cannot decide as to Mr. Witherbee's priority; but the statement leads me to contribute a bit of history, showing an early departure from the "rule-of-thumb" blast-furnace work to the employment of a chemist.

In the autumn of 1868 the department of mining and metallurgy was established at the Rensselaer Polytechnic Institute. In connection with the assay-department a laboratory was equipped for making general commercial analyses.

In January, 1869, Alexander L. Holley, the manager and builder of the Bessemer plant at the Rensselaer Works of John A. Griswold & Co., the first plant erected in the United States under the Bessemer patents, began to send me samples

of ore, pig-iron, and steel for analysis. Early in 1869 I was regularly retained as chemist for the Rensselaer Works; and while the laboratory was not located at the works, for all practical purposes it was the works-laboratory.

The increase in work soon made it necessary for me to secure an assistant. I applied to Prof. Charles F. Chandler, who recommended Dr. August Wendel, who had been an assistant of Rammelsberg and had lately landed in New York. Dr. Wendel remained with me until 1873, when I left Troy. My leaving made necessary the erection of a laboratory at the Rensselaer Works, which was done under my supervision, and Dr. Wendel was employed as chemist.

During the period from 1868 to 1873 my work as consulting mining engineer for John A. Griswold & Co. frequently called me to the Lake Champlain iron-district and involved the examination and sampling of about all the developed mines and prospects, in hunting for Bessemer ores; for, astonishing as the statement may seem, in the early history of the Bessemer process in the United States pig-iron was imported from the West Cumberland district in England. From 1868 to 1872 I frequently visited Mr. Witherbee at Fletcherville, and was always greatly impressed by his energy, originality, and knowledge of the best European practice. My last meeting with him, at Durango, in Mexico, was a pathetic one, for he was then almost totally blind.

On referring to my record of many hundred analyses during those years, I find that samples of ore, pig-iron, slag, and steel came from many widely-scattered works, which would indicate that chemical laboratories were exceptional in connection with iron- and steel-works.

The Behavior of Copper-Matte and Copper-Nickel Matte in the Bessemer Converter.

BY DAVID H. BROWNE, COPPER CLIFF, ONTARIO, CAN.*

(Pittsburg Meeting, March, 1910.)

NICKEL has always been a fruitful mother of problems. Previous to the year 1906 nickel was regarded as an element replacing iron in copper-mattes, and it was believed that the same laws which governed the elimination of iron could be applied to nickel. This conception has lately undergone a complete change, and nickel in all its stages is now regarded as an element replacing copper; in other words, copper-nickel is considered not as two elements, but as one metal.

When we commenced our investigations on Monel metal we made a great many experiments on the conversion of mattes, both high copper and high nickel. We studied closely the composition of the mattes and the changes produced by the blow. We found in all cases certain agreements which we believe are due to some law. Briefly stated, our conclusions are as follows:

1. Nickel is not an element replacing iron in matte.
2. Nickel-copper alloys act in the matte-blow like one metal.
3. Nickel-copper alloys follow, during the matte-blow, exactly the same laws that govern the behavior of copper alone.

This brings up the question: What are the laws governing the production of metallic copper, and what are the relations of copper to sulphur and iron during the Bessemer blow?

* Metallurgist of the Canadian Copper Co.

Our knowledge on this source is derived from the published results of Douglas,[1] Van Liew,[2] Gibb,[3] and Mathewson.[4]

Of these, the paper of Mr. Mathewson is the most complete, and will be made the basis of study. He presents a series of analyses of 10-min. samples from three blows of copper at Anaconda, tabulating these results in the form of curves showing the elimination of iron and sulphur in relation to the blowing-time.

Blowing-time depends on many factors—on the grade of the matte, on the amount of matte in the converter, on the condition of the tuyeres, on the volume and pressure of the blast, on the height of the barometer, and so on. Blowing-time is a variable datum-line. Moreover, Mr. Mathewson's paper was entitled, "Relative Elimination of Iron," etc., while the curves give the "rate of elimination," another matter entirely. Consequently, his curves were not comparable either with each other or with the work of others. They could not be superimposed.

In order to get a fixed datum-line, I recalculated all Mr. Mathewson's analyses and expressed the results in "ratios." Taking the percentage of copper in each analysis as 100, the amounts of iron and sulphur are expressed as percentages of the amount of copper present at that time. For example, a matte containing Cu, 46.08; Fe, 24.30; S, 24.70 per cent. becomes Cu, 46.08, with iron ratio 53.8 and sulphur ratio 54.7. The iron in this matte is 53.8 per cent. of the amount of copper. These ratios were plotted in curves, and these curves were found to compare very closely with each other and with the work of other observers.

Tables I., II., III., and the corresponding Figs. 1, 2, 3, show Mr. Mathewson's analyses as given and as recalculated. Table IV. and Fig. 4 show all these analyses grouped according to 5 per cent. rises in the amount of copper. Fig. 4 expresses, therefore, the average of all Mr. Mathewson's results.

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[1] The Copper Queen Mine, Arizona, *Trans.*, xxix., 511 to 546 (1899).

[2] Relative Elimination of Impurities in Bessemerizing Copper-Matte, *Trans.*, xxxiv., 418 to 421 (1904).

[3] Discussion of the paper of Mr. Van Liew, *Trans.*, xxxiv., 957 to 963 (1894). The Elimination of Impurities During the Process of Making "Best Selected" Copper, *Proceedings of the Institution of Mechanical Engineers*, April, 1895, p. 254.

[4] Relative Elimination of Iron, Sulphur, and Arsenic in Bessemerizing Copper-Mattes, *Trans.*, xxxviii., 154 to 161 (1908).

The curve of iron-elimination is a regular mathematical curve, which can doubtless be expressed in a functional equation. This fact, however, pertains to mathematics and not metallurgy;

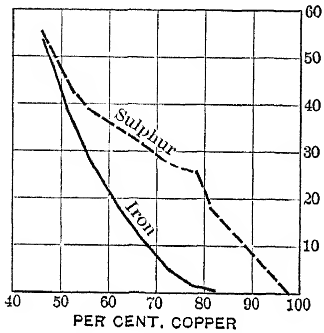


FIG. 1.—MR. MATHEWSON'S TABLE I. ON COPPER BASIS.

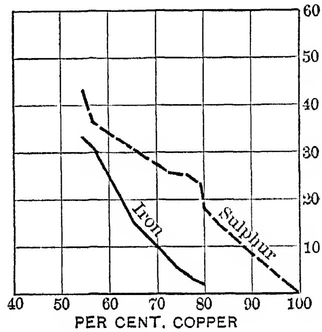


FIG. 2.—MR. MATHEWSON'S TABLE II. ON COPPER BASIS.

for the present it is enough to know that iron slags off with perfect regularity.

On the whole, sulphur disappears in a straight line with a curious jog or reversal occurring when the iron is down to 5 per cent. This will be referred to later.

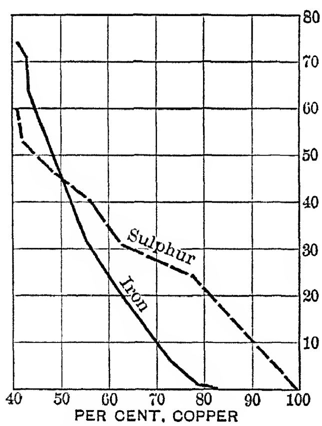


FIG. 3.—MR. MATHEWSON'S TABLE III. ON COPPER BASIS.

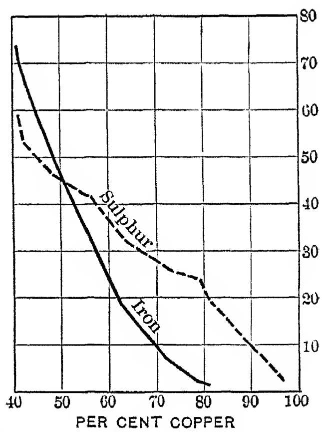


FIG. 4.—MR. MATHEWSON'S TABLES I., II., AND III., COMBINED.

Table V., Fig. 5, and Table VI., Fig. 6, give respectively the results of reverberatory-furnace practice by Mr. Gibb, and Bessemer practice by Mr. Van Liew. The curves are very interesting in showing the parallel nature of the two processes.

Douglas's assays, taken simply of the tap, the white metal, and the blister-copper, do not give a curve, but fix several

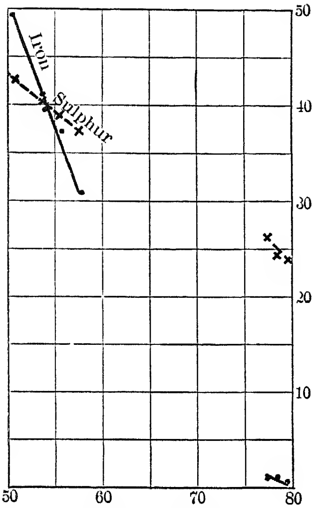


FIG. 4A.—SULPHUR AND IRON RATIOS. DOUGLAS.

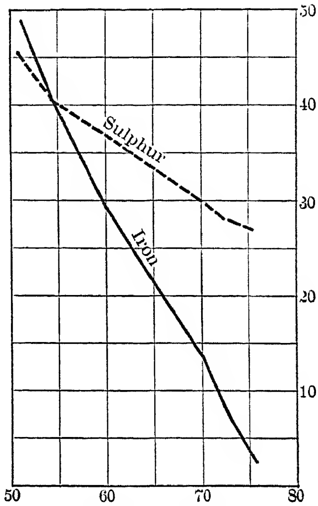


FIG. 5.—SULPHUR AND IRON RATIOS, REVERBERATORY FURNACE. ALLAN GIBB.

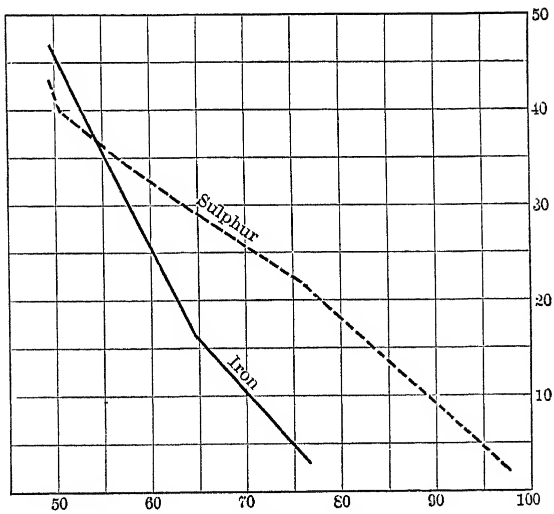


FIG. 6.—SULPHUR AND IRON RATIOS, BESSEMER CONVERTER. W. RANDOLPH VAN LIEW.

valuable reference-points which quite coincide with the curves of all the others, as shown in Fig. 4A. When the results of

these independent observers show such close agreement, it is evident that "relative elimination" is a fixed factor, no matter how much "rate of elimination" or blowing-time may vary. This suggests the idea of the laws governing the action of copper-matte in the Bessemer converter.

Does nickel-matte follow the same laws as copper-matte? This was the problem the Canadian Copper Co. had to solve in 1904. During that year we made very careful tests of a large number of blows, taking samples at intervals of 10 min. during the blow. First we treated matte high in copper and low in nickel, and endeavored to slag the nickel and finish the copper as blister.

Considering the heat of formation, nickel would be expected to follow the iron easily and completely into the slag. Instead of doing so it displays a most extraordinary reluctance to part from the copper, the two metals clinging together in a deathless affinity, so much so that 1 lb. of nickel passing into the slag drags 1.25 lb. of copper with it. Tables VII., VIII., and IX. show the analyses of copper-nickel mattes high in nickel. Tables X., XI., XII., XIII., XIV., and XV. show mattes high in copper. Looking over these data, samples are found of the following composition, which are practically blister-alloy:

	Per Cent.	Per Cent.	Per Cent.
Cu	76.34	84.30	86.90
Ni	21.55	14.35	10.33
CuNi	97.89	98.65	96.23
Fe	0.15	0.40	0.10
S	1.19	0.64	1.04

While nickel during the last stages of the blow does oxidize, yet it lags behind sulphur as a heat-producer. It is impossible to blow all the nickel out. About 2 per cent. is always left in the finished copper.

Tables VII. to XV. show that, even in the most powerful oxidizing conditions we practice, nickel resists oxidation more than its heat-value would indicate.

In attempting to plot these results with copper as a basis no agreement could be observed. Figs. 7A, 8A, 10A, and 15A are given to show how entirely erratic the curves were on this basis. Plotting them on a copper-nickel basis gave remarkable

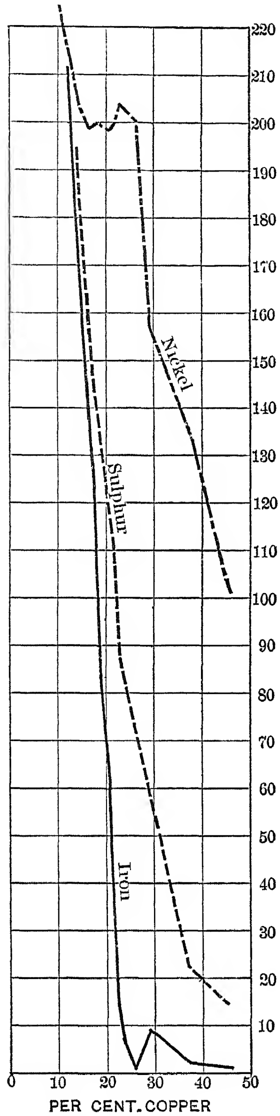


FIG. 7A.—OVERBLOWN COPPER-NICKEL MATTE A. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER = 100.

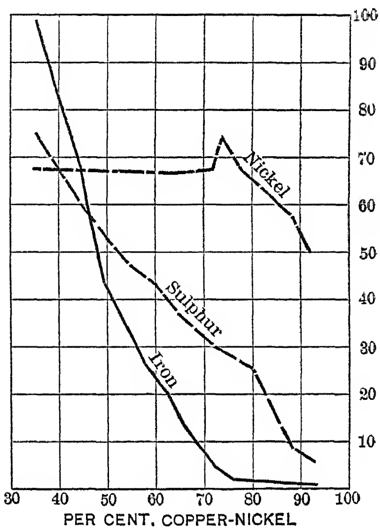


FIG. 7B.—OVERBLOWN COPPER-NICKEL MATTE A. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

results. Taking the percentage of copper and nickel together as 100, and reckoning the ratio of iron and sulphur thereto, we obtained perfectly uniform curves, as shown in Figs. 7B, 8B, 9, 10B, 11, 12, 13, and 14.

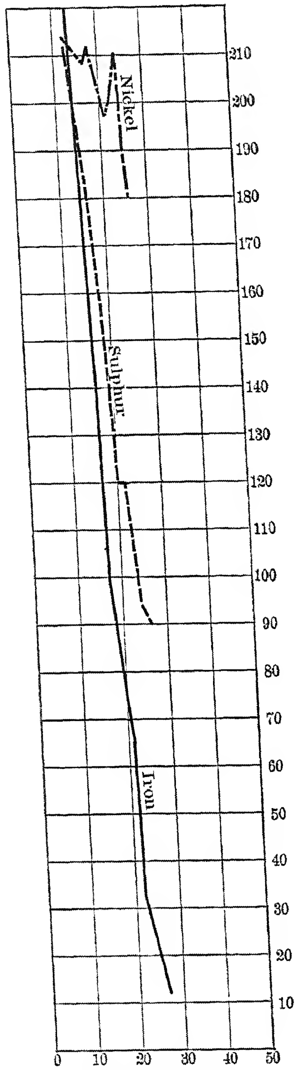


FIG. 8A.—UNDERBLOWN COPPER-NICKEL MATTE B. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER = 100.

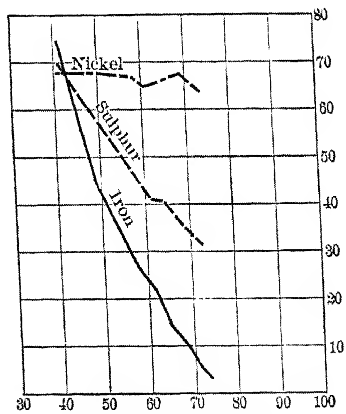


FIG. 8B.—UNDERBLOWN COPPER-NICKEL MATTE B. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

Taking an average of these results, the composition and ratios given in Table XV. are obtained. When these data are plotted, as in Fig. 15B, they yield a set of curves which show beyond a

doubt that copper-nickel is the basis of comparison, and that the elimination of sulphur and iron is a direct function of the copper and nickel together and not of either metal separately. (See Figs. 15A and 15B.) A comparison of our curves obtained in this way with those of Douglas, Gibb, and Van Liew shows a striking parallelism. Their data were, however, somewhat meager, and when Mr. Mathewson, in 1907, published his results we welcomed the opportunity to compare his data with those we had obtained in 1904 and 1906.

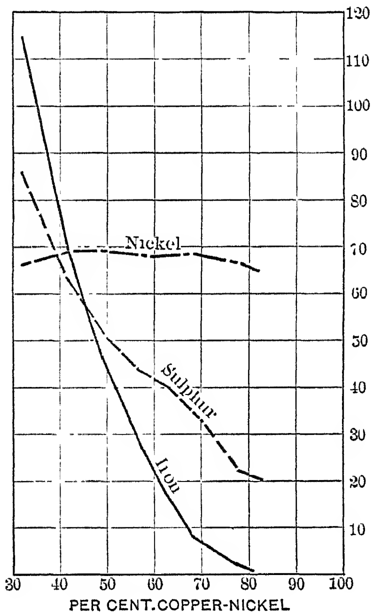


FIG. 9.—ORDINARY COPPER-NICKEL MATTE C. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

As previously stated, there was no agreement whatever to be expected from curves based upon blowing-time. When, however, Mathewson's ratios on a copper basis were compared with our ratios on a copper-nickel basis, the curves were not only similar; they were identical. The iron-curves given in Fig. 4, Mathewson's average on copper-mattes, and our average on copper-nickel mattes, can be superimposed with remarkable accuracy, as shown in Fig. 16A.

As nickel sulphide contains more sulphur than copper sulphide, the sulphur-curves are not so closely comparable (Fig. 16B). They are roughly parallel in the first part of the blow, and coincide after the iron has been eliminated. A curious point of resemblance is the little jog or reversal in the sulphur-

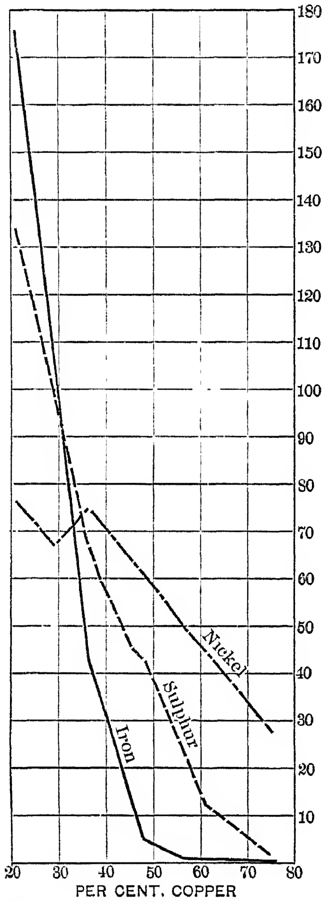


FIG. 10A.—COPPER-NICKEL MATTE D. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER = 100.

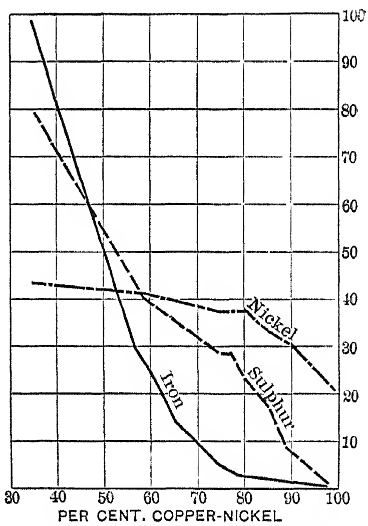


FIG. 10B.—COPPER-NICKEL MATTE D. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

curve, which occurs when the iron is reduced to 5 per cent. All the curves of individual blows show this. The only explanation I can offer is that at this point a certain amount of metallic copper or copper-nickel separates out as metal, and, falling to the bottom of the converter, leaves the sulphur with

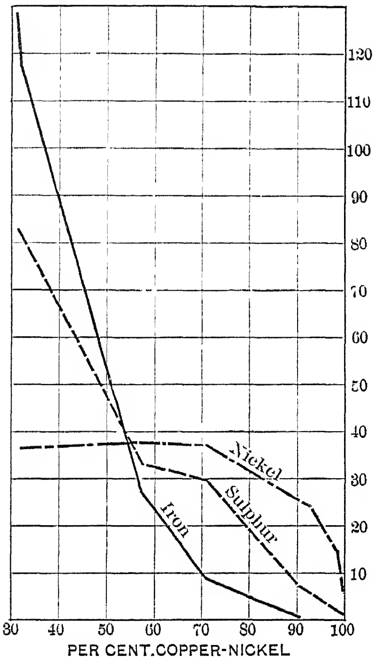


FIG. 11.—COPPER-NICKEL MATTE E. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

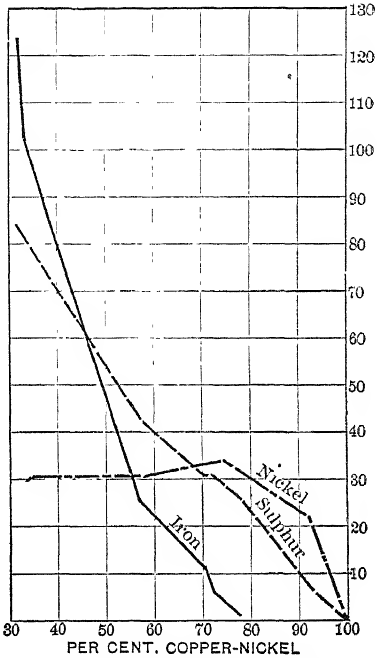


FIG. 13.—COPPER-NICKEL MATTE G. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

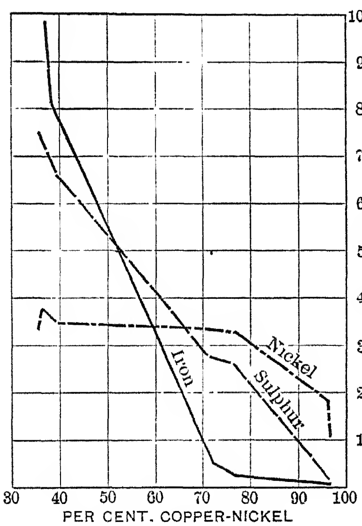


FIG. 12.—COPPER-NICKEL MATTE F. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

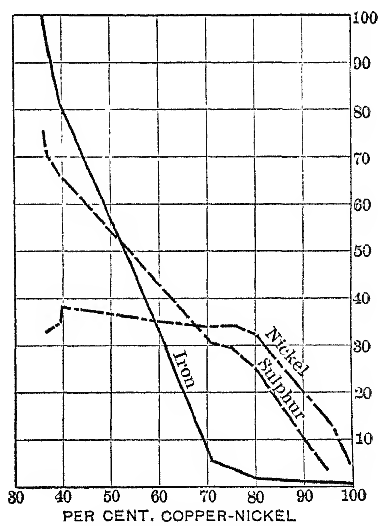


FIG. 14.—COPPER-NICKEL MATTE H. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

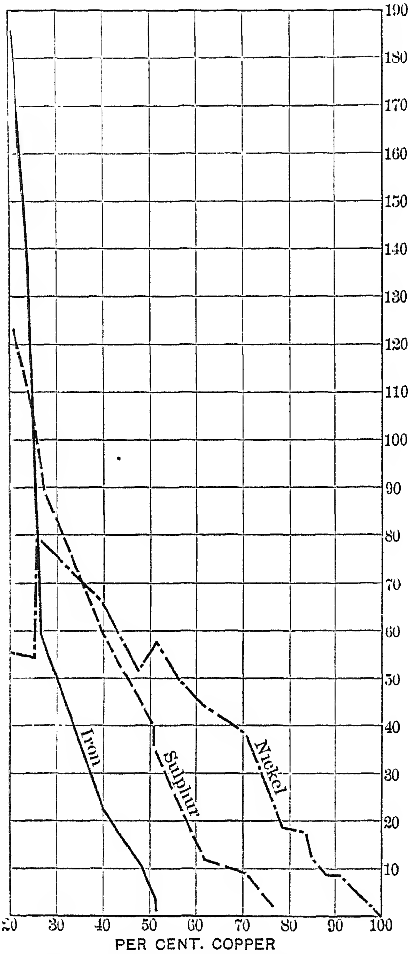


FIG. 15A.—COPPER-NICKEL MATTES D, E, F, G, AND H. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER = 100.

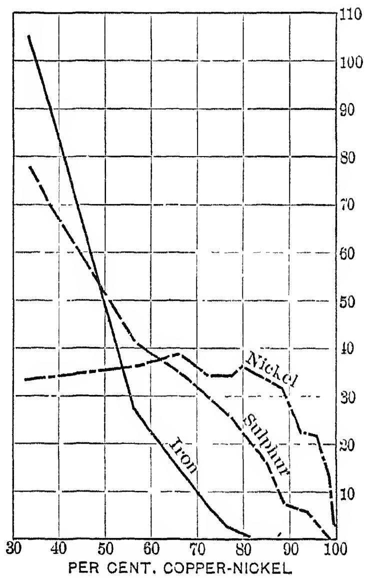
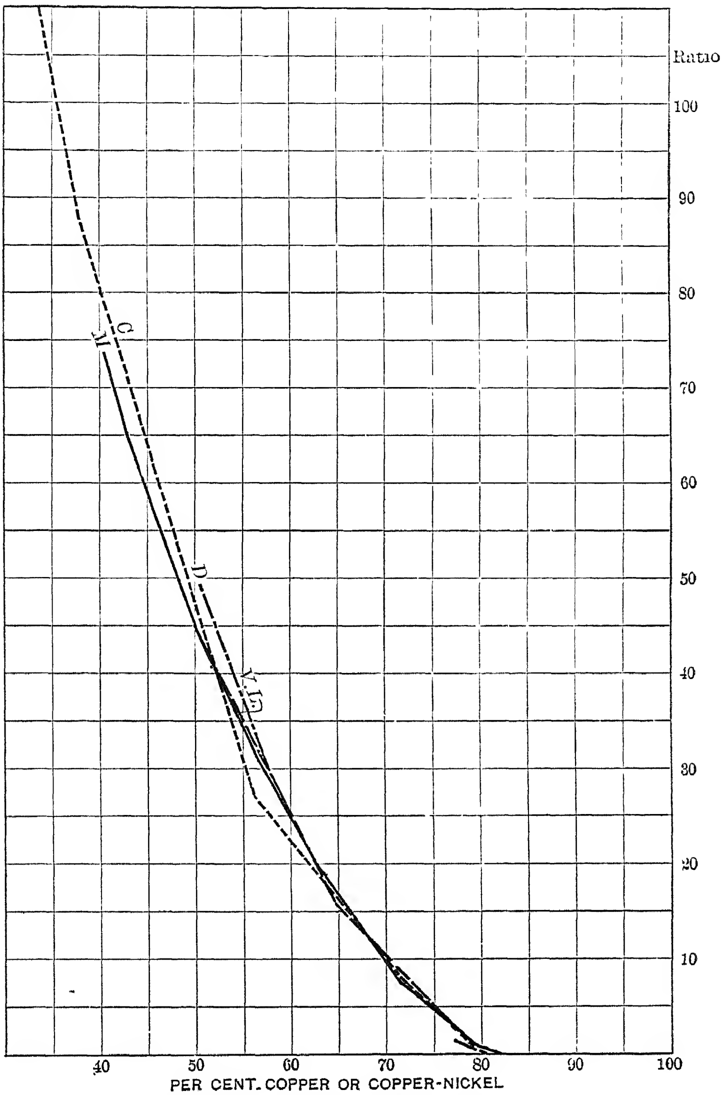


FIG. 15B.—COPPER-NICKEL MATTES D, E, F, G, AND H. RATIOS OF NICKEL, IRON, AND SULPHUR. COPPER-NICKEL = 100.

which it was united to be absorbed by the overlying matte, which makes the matte at this point higher in sulphur than it was before the metal separated out. In order to clear up this matter, I have calculated the sulphur required to form Cu_2S and FeS in all the Anaconda mattes, and the sulphur required to form Cu_2S, Ni_2S, and FeS in our own mattes. After tabulating these results, and deducting the sulphur actually present, the deficiency of sulphur at any point is given. Our own mattes show,

Fig. 18, that the sulphur-deficiency decreases, and between 50 and 75 per cent. of copper-nickel the sulphur is in excess of that

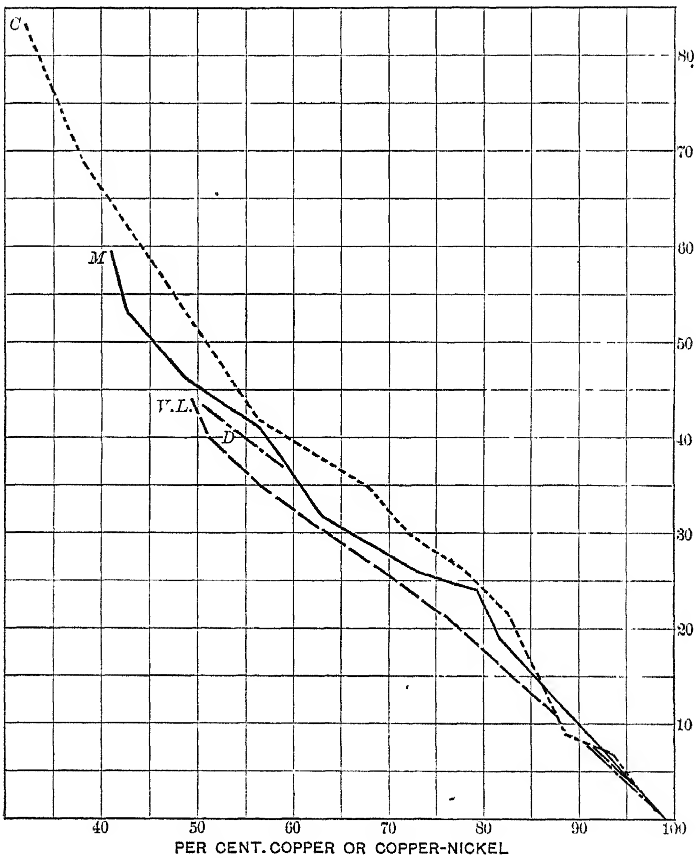


C. Canadian Copper Co. D. Douglas. M. Mathewson. V. L. Van Liew.

Fig. 16A.—Iron Ratios.

required to unite with the elements. The Anaconda blows, Fig. 17, do not show an excess of sulphur at any point, but they

show a smaller deficiency of sulphur between 55 and 75 per cent. of copper than at other points during the blow, a result which shows that there is the same tendency in both copper- and copper-nickel mattes. An excess of sulphur, or a lessened



C. Canadian Copper Co. D. Douglas. M. Mathewson. V. L. Van Liew.

FIG. 16B.—SULPHUR RATIOS.

deficiency of sulphur, both point to the dropping-out of some metal or alloy in metallic form.

All these coincidences seem to prove that copper-nickel alloys form, as far as conversion is concerned, one homogeneous metal. They act together—no matter what the proportion be; high copper and low nickel, or *vice versâ*, they present in

conversion the same curious resistance to oxidation, and their relations towards iron are absolutely similar to the relations of

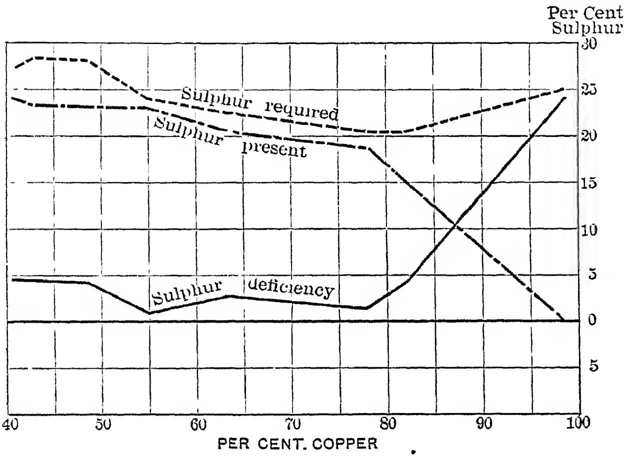


FIG. 17.—SULPHUR-RELATIONS OF COPPER-MATTES.

copper alone. Of all such alloys, Monel metal is the one we know most about at present.

This alloy contains about 70 per cent. of nickel and 30 per

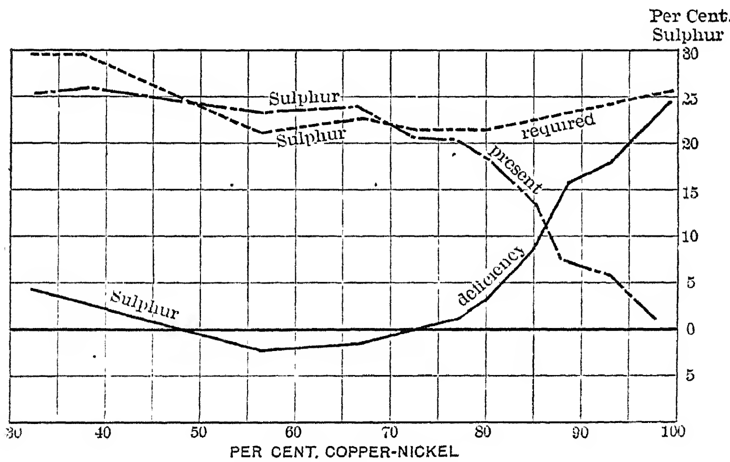


FIG. 18.—SULPHUR-RELATIONS OF COPPER-NICKEL MATTES.

cent. of copper. This proportion is very nearly that in which the two metals exist in the ores of the Canadian Copper Co., and by careful attention to the furnace-charge a Bessemer matte can be produced within 1 per cent. of the required proportion.

The matte is blown till the iron is eliminated, the sulphur is removed by roasting, and the combined oxides reduced to metal. As the compound metals have never been separated from each other, the particles of each seem to be in more intimate contact than can be attained by any synthetic method of manufacture. It seems incredible, but it is a fact, that we have not been able to produce by melting copper and nickel together an alloy having the same physical properties as the alloy produced direct from the matte.

There seems to be some physical reason why nickel and copper cling together in this way, and this resistance to oxidation in the converter is in some way connected with the resistance of the finished alloy to oxidation and corrosion, but what this reason is we do not at present know. A comparison of the properties of copper, nickel, and Monel metal may be of interest.

	Copper, Rolled	Nickel, Rolled.	Monel Metal.		
			Cast.	Rolled.	Annealed.
Tensile strength, lb. per sq. in.,	34,000	75,500	85,000	100,000	110,000
Elastic limit, lb. per sq. in., .	18,000	21,000	40,000	50,000	80,000
Elongation in 2 in., per cent.. .	52	43.9	25	30	25
Contraction, per cent., . . .	57	57	25	50	50
Melting-point, °C., ...	1,084	1,500	1,360		

As before mentioned, the laws governing copper-nickel alloys are not thoroughly understood. The present paper has been in preparation for five years, and is now offered, even though it is imperfect and incomplete, at the request of Mr. Mathewson and others, in the hope that further research may elucidate the thermo-chemical problems involved.

TABLE I.—*Recalculation of Mathewson's Data, Given in His Table I.*

Composition			Ratios		
Cu Per Cent	Fe Per Cent.	S. Per Cent	Cu = 100. Per Cent.	Fe. Per Cent	S. Per Cent.
46.08	24.30	24.70	46.08	52.7	53.5
46.02	23.70	22.95	46.02	51.5	49.8
51.46	20.50	23.10	51.46	39.8	44.8
53.27	18.70	22.15	53.27	35.0	41.5
56.29	16.20	21.85	56.29	28.7	38.6
59.90	13.70	21.95	59.90	22.8	36.6
62.67	11.40	21.35	62.67	18.2	34.2
67.89	7.60	21.15	67.89	11.2	31.2
73.97	3.40	20.10	73.97	4.6	27.2
77.82	0.90	19.60	77.82	1.5	25.2
74.16	2.60	16.60	74.16	3.5	22.4
81.72	0.20	15.35	81.72	0.3	18.8
98.50	tr.	0.78	98.50	tr.	0.8
98.57	0.01	0.78		(See Fig. 1.)	
99.08	tr.	0 01			

TABLE II.—*Recalculation of Mathewson's Data, Given in His Table II.*

Composition.			Ratios.		
Cu. Per Cent.	Fe. Per Cent.	S. Per Cent.	Cu = 100. Per Cent	Fe. Per Cent.	S. Per Cent.
54.81	18.00	23.00	54.81	32.8	41.9
56.26	17.40	20.30	56.26	31.0	35.9
38.25	30.10	14.20	38.25	78.6	37.3
64.71	10.40	20.10	64.71	15.2	31.1
68.86	7.40	19.50	68.86	10.7	28.5
72.72	4.20	18.20	72.72	5.8	25.1
78.29	1.60	19.50	78.29	2.04	24.9
79.49	0.50	18.60	79.49	0.63	23.4
78.27	0.90	16.40	78.27	1.13	20.9
79.63	1.00	14.90	79.63	1.25	18.7
82.85	0.80	13.10	82.85	0.9	15.8
98.46	0.019	0.73	98.46	......	0.8
98.75	0.012	0.53	98.75	......	0.6
99.26	0.004	tr.	99.26		
			(See Fig. 2.)		

TABLE III.—*Recalculation of Mathewson's Data, Given in His Table III.*

Composition.			Ratios.		
Cu. Per Cent.	Fe Per Cent	S Per Cent.	Cu=100 Per Cent	Fe Per Cent.	S Per Cent.
40.68	29.60	24.30	40.68	72 7	59.7
41.11	29.10	23.50	41.11	70.7	57.2
42.50	27.80	22 70	42.50	65.5	53.4
49.16	23.70	22.60	49.16	48.2	45.9
55.86	17.50	22.90	55 86	31.4	41.1
62.97	12.10	20.40	62.97	19.3	32.4
72.06	5 30	19.10	72.06	7.35	26.5
78.96	0.90	18.70	78 96	1.14	23.7
81.32	0.40	15 90	81.32	0.5	19.5
98.24	0.011	0.81	98.24	tr.	0.8
98.56	0.008	0 86	98.56		
99.17	tr.	0 002	99.17	(See Fig. 3.)	

TABLE IV.—*Recalculation of Mathewson's Data, Combined from His Tables I., II., and III.*

Average Composition.			
Cu Per Cent.	Fe. Per Cent.	S. Per Cent.	
41.33	28.41	23.5	
47.41	23.9	23.4	
51.18	19.06	22.75	
57.08	16.20	21.75	
63.85	11.03	20.61	
68.37	7.50	20.32	
73.22	3.87	18.50	
78.84	0.98	18.28	
81.73	0.46	14.78	
98.51	tr.	0.5	(See Fig. 4.)

NOTE.—Average taken of all analyses with Cu 40 to 45, 45 to 50, etc.

TABLE V.—*Recalculation of Gibb's Data of Reverberatory-Furnace Practice.*

Composition.			Ratios.		
Cu. Per Cent.	Fe. Per Cent.	S. Per Cent	Cu=100 Per Cent.	Fe. Per Cent.	S. Per Cent.
50.7	25.1	23.3	50.7	49.5	45.9
54.0	22.1	22.1	54.0	40.9	40.9
60.0	17.1	22.1	60.0	28.5	36.8
69.8	9.2	20.9	69.8	13.10	30.0
72.7	5.0	20.5	72.7	6.80	28.0
75.4	1.8	20.4	75.4	2.40	27.0

(See Fig. 5.)

TABLE VI.—*Recalculation of Van Liew's Data of Bessemer-Converter Practice.*

Composition			Ratios.		
Cu. Per Cent	Fe. Per Cent.	S Per Cent.	Cu=100. Per Cent.	Fe. Per Cent.	S. Per Cent.
49.72	23.31	21.28	49.72	46.5	43.0
50.20	23.15	20.99	50.20	46.1	40.0
56.88	17.85	19.74	56.88	31.40	34.60
64.60	10.50	18.83	64.60	16.30	29.0
76.37	2.40	16.30	76.37	3.10	21.30
99.12	0.038	0.15	99.12	0.03	0.15

(See Fig. 6.)

TABLE VII.—*Composition of Copper-Nickel Matte A. High Nickel.*

(Sampled at 10-min. intervals during the blow.)

Composition					Ratios.				Ratios			
Cu Per Cent.	Ni. Per Cent.	CuNi. Per Cent.	Fe. Per Cent	S Per Cent	Cu = 100. Per Cent.	Ni. Per Cent.	Fe. Per Cent.	S Per Cent.	CuNi = 100. Per Cent.	Ni. Per Cent.	Fe. Per Cent.	S. Per Cent.
11.40	23.72	35.12	34.75	26.54	11.40	224.4	316	249	35.12	67.6	98.9	75.6
14.30	29.04	43.34	26 27	26 68	14.30	203	184	186	43.34	67.0	60.8	61 6
16.55	32.96	49.51	21.20	26.62	16.55	199	128	161	49.51	66.7	42.8	53 7
18.85	37.54	56.39	15.61	26 02	18.85	200	83	138	56.39	66.3	27.6	46.0
20 60	40.46	61.06	12.34	25.68	20.60	196	59.8	124	61.06	66.2	20.2	42.1
21.65	43.32	64 97	8.87	23.88	21.65	200	41	110	64.96	66.7	13.7	36 8
22.95	50.64	73.59	3.11	21.72	22.95	231	13.5	94.7	73.59	68.7	4 2	29.4
20.20	54.38	74.58	2.75	20.18	20.20	261	13.7	100	74.58	73	3.7	27 1
23.60	53.36	76.96	1.68	20.70	23.60	226	6.8	87.3	76.96	69.2	2.2	26.8
26.30	52.48	78.78	0.45	20.18	26.30	200	1.71	76.6	78.78	66.5	0.5	25 6
29.30	46.12	75 42	2 65	20.98	29.30	157	9.0	71.6	75.42	61.2	3.4	27.8
37.05	50.98	88.03	0.96	8 38	37 05	137	2.6	22.6	88.03	58	1.1	9.5
45.90	46.50	92.40	0.56	6 52	45.90	101	1.2	14.2	92.40	49.9	0.6	7.0
					(See Fig 7A.)				(See Fig. 7B)			

TABLE VIII.—*Composition of Copper-Nickel Matte B. High Nickel. (Sulphur by Difference.)*

(Sampled at 10-min. intervals during the blow.)

Composition.					Ratios				Ratios.			
Cu. Per Cent.	Ni. Per Cent.	CuNi. Per Cent.	Fe. Per Cent.	S. Per Cent.	Cu =100. Per Cent.	Ni. Per Cent.	Fe. Per Cent.	S. Per Cent.	CuNi =100. Per Cent.	Ni. Per Cent.	Fe. Per Cent.	S. Per Cent.
12.93	27.60	40.53	30.5	27.6	12.93	213	235	211	40.53	68	75.2	68.0
15.66	33.00	48.66	22.54	28.89	15.66	210	143	184	48.66	67.9	46.3	59.7
17.32	36.02	53.34	19.0	27.66	17.32	209	109	159	53.34	67.3	35.6	52.7
18.08	38 26	56.34	16.35	27.31	18.08	211	90	151	56.34	67.5	27.1	48.4
20.86	40.12	60.98	13.75	25.23	20 86	197	65	120	60.98	65.8	22.4	41.1
21.41	42 98	64.39	9.90	25.73	21.41	200	46	120	64.39	66.7	15.7	39.8
22.02	46.66	68 68	7.40	24.92	22.02	211	33	113	68.68	67.9	10.8	36.4
24.80	47.14	71.94	4.50	23.56	24.80	190	18	95	71.94	65.3	6.24	32.6
26 26	46.90	73.16	3.15	23.69	26 26	178	12	90	73.16	64	4.3	32 4
					(See Fig. 8A.)				(See Fig. 8B.)			

TABLE IX.—*Composition of Copper-Nickel Matte C. High Nickel.*

(Sampled at 10-min. intervals during the blow.)

Composition					Ratios			
Cu Per Cent.	Ni. Per Cent.	CuNi Per Cent.	Fe Per Cent.	S Per Cent	CuNi = 100 Per Cent.	Ni Per Cent.	Fe Per Cent.	S. Per Cent
10.55	21.60	32.15	37.25	27.70	32.15	67.2	11.58	86.2
12.80	28.50	41.30	28.60	26.25	41.30	69	69.2	63.5
15.40	34.00	49.40	21.70	25.95	49.40	68.9	43.9	52.5
18.10	39.80	57.90	14.95	25.95	57.90	68.3	25.8	44.5
19.80	43.25	63.05	10.30	25.10	63.05	68.6	16.3	39.8
21.60	46.85	68.45	5.57	25.04	68.45	68.4	8.1	35.1
25.90	52.20	78.10	2.15	17.70	78.10	66.9	2.7	22.6
28.10	53.4	81.5	1.07	16.10	81.5	65.5	1.3	20.0
						(See Fig. 9.)		

TABLE X.—*Composition of Copper-Nickel Matte D. High Copper.*

(Sampled at 10-min. intervals during the blow.)

	Composition.					Ratios				Ratios			
	Cu. Per Ct	Ni Per Ct.	CuNi. Per Ct.	Fe. Per Ct.	S. Per Ct.	Cu = 100. Per Ct.	Ni. Per Ct.	Fe. Per Ct.	S Per Ct.	CuNi = 100. Per Ct.	Ni. Per Ct.	Fe. Per Ct.	S Per Ct.
Tap	20.15	15.65	35.80	32.20	26.92	20.15	76.0	175	134	35.80	43.7	98	77.7
Con..	39.72	26.75	66.47	8.95	23.84	39.72	66.3	22	60	66.47	39.6	13.2	35.4
Tap..	20.75	13.70	34.45	36.0	27.10	20.75	66.0	173	131				
Con..	36.04	23.00	59.04	15.67	24.71	36.04	74.2	43.6	68.8	59.04	40.7	26.2	41.8
Con..	47.96	29.13	77.09	2.45	20.74	47.96	60.8	5.0	43.2	77.09	37.8	3.18	26.9
Tap .	21.65	12.40	34.05	37.00	27.08	21.65	57.1	171	125				
Con.	35.34	21.15	56.49	16.80	25.15	35.34	59.3	47.2	70.8	56.49	37.7	29.7	44.4
Con..	47.26	28.25	75.51	3.22	21.43	47.26	59.9	6.8	45.2	75.51	37.4	4.24	28.3
Con..	51.23	29.51	80.74	0.55	18.32	51.23	57.6	1.0	35.6	80.74	37.2	0.68	22.6
Con..	56.40	28.49	84.89	0.47	13.31	56.40	50.3	0.83	24.4	84.89	33.5	0.55	16.2
Con..	61.22	27.77	88.99	0.25	7.60	61.22	45.1	0.40	12.4	88.99	31.1	0.3	8.6
Con.	76.34	21.55	97.89	0.15	1.19	76.34	28.1	0.20	1.56	97.89	22.0	tr.	1.2
							(See Fig 10A.)				(See Fig 10B.)		

TABLE XI.—*Composition of Copper-Nickel Matte E. High Copper.*

	Composition.					Ratios.				Ratios			
	Cu. Per Cent.	Ni. Per Cent.	CuNi. Per Cent.	Fe. Per Cent.	S. Per Cent.	Cu= 100. Per Cent.	Ni. Per Cent.	Fe. Per Cent.	S. Per Cent.	CuNi =100. Per Ct.	Ni. Per Ct.	Fe. Per Ct.	S. Per Ct.
Tap	20.60	11.80	32.4	38.35	27.36	20.60	57.25	186	133	32.4	36.7	118	84.5
Tap.	20.30	10.85	31.15	39.5	27.4	20.30	53.3	194	135	31.15	34.8	127	81.8
Tap	20.80	12.05	32.85	......	......	20.80	57.4	......	....	32.85	36.8		
Con..	36.05	21.45	57.50	15.20	19.55	36.05	59.5	42.15	53.1	57.50	37.4	26.4	34
Con.	45.28	26.15	71.43	5.50	21.24	45.28	57.7	12.15	46.9	71.43	36.8	7.7	29.8
Con.	70.20	22.60	92.80	0.20	7.15	70.20	32.2	0.28	10.17	92.80	24.4	0.22	7.7
Con..	84.30	14.35	98.65	0.40	0.64	84.30	17.0	0.47	0.76	98.65	14.5	0.41	0.7
Con	90.90	8.00	98.90	0.10	1.18	...	......	..	.	98.90	8.1	tr.	1.2
											(See Fig. 11.)		

TABLE XII.—*Composition of Copper-Nickel Matte F. High Copper.*

(Sampled at 10-min. intervals during the blow.)

	Composition.					Ratios.				Ratios.			
	Cu. Per Cent	Ni. Per Cent.	CuNi. Per Cent.	Fe Per Cent.	S Per Cent	Cu=100. Per Cent	Ni Per Cent.	Fe. Per Cent	S Per Cent.	CuNi =100. Per Ct.	Ni Per Ct.	Fe Per Ct.	S. Per Ct
Tap	25.70	13.57	39.27	32.45	26.15	25.70	52.7	126.1	101.3	39.27	34.8	82.5	66.5
Con	47.15	23.32	70.47	5.30	21.05	47.15	50.3	11.4	45.4	70.47	33.1	7.5	28.5
Tap	24.80	13.85	38.65	33.0	26.12	24.80	55.7	132.5	104.7	38.65	36.0	85.5	68
Con	47.35	23.32	70.67	6.0	20.90	47.35	49.2	12.7	44.0	70.67	33.0	8.5	29.8
Tap	24.10	12.25	36.35	35.20	26.56	24.10	50.9	146.2	110.5	36.35	33.5	98.0	73.0
Con	49.9	23.7	73.6	4.3	22.26	49.9	47.5	8.8	44.5	73.6	32.7	5.9	31.1
Con	51.95	25.0	76.95	1.97	19.64	51.95	48.0	3.78	37.75	76.95	32.4	3.6	26.4
Con	78.32	18.77	97.09	0.65	2.21	78.32	23.9	0.83	2.82	97.09	19.4	0.7	2.2
Con	88.7	7.78	96.48	0.55	1.10	88.7	8.79	0.62	1.13	96.48	8.0	0.6	1.2
Con	86.90	10.33	97.23	0.10	1.04					97.23	10.6	0.1	1.1

(See Fig. 12.)

TABLE XIII.—*Composition of Copper-Nickel Matte G. High Copper.*

(Sampled at 10-min. intervals during the blow.)

	Composition.					Ratios.				Ratios.			
	Cu Per Cent	Ni Per Cent	CuNi. Per Cent	Fe Per Cent	S Per Cent	Cu=100. Per Cent.	Ni. Per Cent.	Fe. Per Cent.	S. Per Cent.	CuNi =100. Per Ct	Ni. Per Ct.	Fe. Per Ct	S Per Ct.
Tap	23.10	10.13	33.23	37.80	26.74	23.10	43.7	164	116	33.23	30.8	113	80.5
Con	46.60	23.45	70.05	7.26	22.28	46.60	50.3	15.5	49.2	70.05	33.5	103	31.8
Tap	22.60	9.26	31.86	38.8	26.6	22.60	41.0	172	118	31.86	29.1	122	83.5
Con	39.20	17.70	56.90	14.95	24.42	39.20	42.6	38.1	62.2	56.90	31.2	26.6	43
Con	47.80	24.15	71.95	5.20	22.50	47.80	50.3	10.87	47.1	71.95	33.9	7.2	31.6
Con	53.75	24.80	78.55	0.75	20.26	53.75	46.3	1.39	42.0	78.55	31.6	0.9	25.8
Con	72.72	20.95	93.67	0.11	5.65	72.72	30.2	0.15	7.78	93.67	22.4	0.13	6.0
Con	98.30	0.85	99.15	0.12	0.51	98.30	0.86	0.12	0.52	99.15	1.1	0.12	0.5

(See Fig. 13.)

TABLE XIV.—*Composition of Copper-Nickel Matte H. High Copper.*

(Sampled at 10-min. intervals during the blow.)

	Composition					Ratios.				Ratios.			
	Cu Per Cent.	Ni. Per Cent.	CuNi. Per Cent.	Fe. Per Cent.	S. Per Cent.	Cu =100 Per Cent.	Ni. Per Cent.	Fe. Per Cent	S. Per Cent.	CuNi =100. Per Ct.	Ni Per Ct.	Fe Per Ct	S. Per Ct.
Tap	25.20	14.80	40.0	32.0	26.0	25.20	58.7	126.9	103.2	40.0	37	80	65
Con	46.95	24.46	71.41	4.90	21.76	46.95	52.1	10.4	46.3	71.41	34.4	6.8	30.5
Tap	25.80	13.79	39.59	32.25	26.10	25.80	53.2	124.6	100.8	39.59	35	82	66
Con	49.95	24.74	74.69	4.0	21.74	49.95	49.3	7.99	43.37	74.69	33.2	5.3	29
Tap	24.80	12.64	37.44	34.25	26.25	24.80	50.8	137.8	103.6	37.44	33	92	70
Con	48.75	25.89	74.64	3.57	21.15	48.75	51.0	7.3	43.3	74.64	34.8	4.8	28.5
Tap	23.90	11.87	35.77	35.70	26.62	23.90	49.6	148.8	111.2	35.77	33.5	100	75
Con	54.50	25.12	79.62	1.10	20.80	54.50	46.10	2.02	56.7	79.62	32	1.4	25.2
Con	84.90	11.75	96.65	0.61	0.95	84.90	13.8	0.72	1.12	96.65	12.3	0.7	1.0
Con	92.3	4.65	96.95	0.26	0.49								
Con	97.38	1.79	99.17	0.48	0.33	97.38	1.84	0.49	0.34				

(See Fig. 14.)

TABLE XV.—*Composition of Copper-Nickel Mattes D, E, F, G, H, Combined. High Copper.*

Composition.					Ratios.				Ratios.			
Cu. Per Cent.	Ni. Per Cent.	CuNi. Per Cent.	Fe Per Cent.	S. Per Cent	Cu = 100. Per Cent.	Ni. Per Cent.	Fe Per Cent.	S. Per Cent.	CuNi = 100 Per Cent.	Ni. Per Cent.	Fe Per Cent.	S Per Cent.
20.50	11.36	31 86	37.91	25.42	20.50	55	185	124	31.86	34 40	115	77
24.56	13.30	37.86	33.76	26.34	24 56	54	133	107	37.86	35.20	91 7	69 3
26.19	20.82	57.01	15.56	23.46	26.19	79	59	89	57.01	37.2	27 9	41 9
40 72	26.75	67.47	8.95	23.84	40 72	65	22	58	67.47	39.6	13.2	35.4
47.64	24.32	71 96	5 06	21.72	47.64	51	10	46	71.96	33.8	7.02	30.2
51.08	26.46	77.54	1.90	20.34	51.08	51	3	39	77.54	34 1	2.45	26 3
51.23	29.51	80.74	0.55	18 32	51.23	57	1	35	80.74	36.6	0.68	22 7
56.40	28.49	84.89	0.47	13.81	56 40	50	..	24	84.89	33.6	0.55	16.25
61.22	27.77	88.99	0.25	7.60	61 22	45	..	12	88.99	31.2	0.28	8.52
71.46	21.77	93.23	0.16	6.40	71.46	38		9	93.23	23.4	0 17	6 87
76.34	21.55	97.89	0.15	1.19	76 34	28	..	1.5	97.89	21.9	0.15	2.12
84.30	14 35	98 65	0.40	0 64	84 30	17	.	...	98.65	14.5	0.41	0.65
91.10	7.65	98.75	0.10	1.37	91.10	8	...	.. .	98.75	7.72	0 10	1.38
78.22	18.77	97.99	0.65	2.21	78.22	24	...	...	97.99	19.07	0.67	2.26
88.70	7.78	96.48	0 55	1.00	88.70	8	...	...	96.48	8.06	0.56	1.02
88.29	8.72	97.01	0 71	1.17	88.29	9	.	...	97.01	9 0	0.73	1.21
84 90	11.75	96.65	0.61	0.95	84.90	13	..	...	96.65	12.5	0.63	0.98
98.30	0.85	99.15	0.12	0.51	98.30	0.9	...	....	99.15	0.86	0.12	0.52
97.38	1.79	99.17	0 48	0.33	97.38	1.8	...	.....	99.17	1.79	0.48	0.33
					(See Fig. 15A.)				(See Fig. 15B.)			

The Behavior of Copper-Slags in the Electric Furnace.

BY LEWIS T. WRIGHT, SAN FRANCISCO, CAL.

(Pittsburg Meeting, March, 1910.)

I HAVE long been aware that ferruginous copper furnace-slags if fused in the electric arc will yield metallic iron containing copper, and in order to confirm this knowledge, and to obtain further information of the behavior of such slags in the high temperature and reducing conditions of the electric furnace, a number of samples of slag were fused in an experimental electric furnace.

The results obtained, given in Table I., show that a very large amount of metallic iron was produced, and that this iron carried with it practically all the copper, gold, and silver, and more than 50 per cent. of the sulphur, of the original slag.

The metallic buttons formed in the electric furnace were clean, hard, but not brittle.

TABLE I.—*Results of Fusing Ferruginous Copper-Slags in an Electric Furnace.*

Experiment No.	9.	11.	12.	13.	14	15.	16.
Slag used, grams	700	700	900	700	700	700	700
Strength of current, amperes	75	70	70	80	80	75	75
Time of fusion, hours	1–20′	1–20′	1–15′	1–15′	1	1–20′	1–15′
Weight of button, grams	186.0	213.78	218.95	216.32	162.82	166.9	153.45
Concentration, per cent	26.57	30.54	24.32	30.91	23.26	23.84	21.92
Analysis of Slag Used:							
Cu, per cent	0.55	0.25	0.30	0.28	0.28	0.45	0.38
SiO_2, per cent	36.8	39.1	39.2	39.6	39.6	45.38	38.48
Fe, per cent	42.3	38.8	38.8	40.1	40.1	40.3	38.3
S, per cent	......	......	......	......	......	1.7	1.47
Ag, ounces per ton	0.29	0.3	0.27	0.29	0.29	0.33	0.18
Au, dollars per ton	0.40	0.30	0.30	0.30	0.30	0.30	0.15
Analysis of Iron Button:							
Cu, per cent	1.62	0.84	1.09	0.82	1.4	1.36	1.62
SiO_2, per cent	0.06	0.46	0.12	0.48	0.6	0.5	0.2
Fe, per cent	90.91	92.7	94.52	88.0	91.06	93.39	94.28
S, per cent	4.4	2.25	3.96	1.86	2.9	3.91	3.9
Ag, ounces per ton	1.4	0.63	0.62	1.11	0.69	0.58	0.48
Au, dollars per ton	0.80	0.40	0.40	0.50	0.60	1.20	0.80
Analysis of Slag After Fusion:							
Cu, per cent	0.1	0.05	0.05	0.05	0.05	trace	trace
Fe, per cent	18.8	6.5	4.9	5.0	6.3	19.31	17.23
S, per cent	0.4	0.62	0.19	0.22	0.23	0.47	0.55
SiO_2, per cent	51.38	59.0	55.8	63.5	53.8	51.12	51.84
Ag, ounces per ton	0.025	trace	0.02	0.0075	0.025	trace	trace
Au	trace	trace	trace	trace	trace	trace	trace

Sampling Anode-Copper, with Special Reference to Silver-Content.

BY WILLIAM WRAITH, ANACONDA, MONT.

(Pittsburg Meeting, March, 1910.)

I. Introduction.

At the Washoe smelter, Anaconda, Mont., the blister-copper from the converters is transferred, by means of a crane, to a refining-furnace, in which it is brought to proper pitch by means of air and poles. From this furnace the refined copper, containing gold and silver, is poured into cast-iron molds placed on a platform-conveyor, giving an anode 36.75 in. long, 28 in. wide, and $2\frac{1}{16}$ in. thick.

By reason of a disagreement between the smelter and the refinery, in reference to the silver-content of the anodes, it became necessary to investigate the methods of sampling used.

II. Smelter Method of Sampling.

At the smelter the sampling is done by "shotting" into water a small portion of the molten stream of copper as it flows from the furnace, by "batting" the stream with a wooden paddle. The first sample is taken 30 min. after the pouring is started, three other samples being taken at 1-hr. intervals, each sample weighing from 4 to 6 oz. The samples are dried, examined for particles of burnt wood, screened on a 10-mesh screen of No. 8 wire to remove the fines, the oversize then screened on a 4-mesh screen of No. 20 wire to remove the coarse, the undersize of this screen being taken as the sample. The four portions thus obtained are thoroughly mixed and split in half by passing over a 16-slot splitting-device, slots being 0.5 in. wide. One-half of the sample is sent to the laboratory for analysis and one-half kept as a duplicate.

III. Refinery Method of Sampling.

The method used at the refinery and also used in this investigation is the one developed by Dr. Edward Keller, of

Perth Amboy, N. J., in which every fourth anode of a lot is drilled, using a 99-hole template, illustrated in Fig. 1, which also shows the shape of the anode. The holes of the template were used in continuous order, one hole to the anode; for example, if in lot No. 890, holes from No. 1 to No. 90 were used, then in lot No. 891, the sampling would commence with hole No. 91.

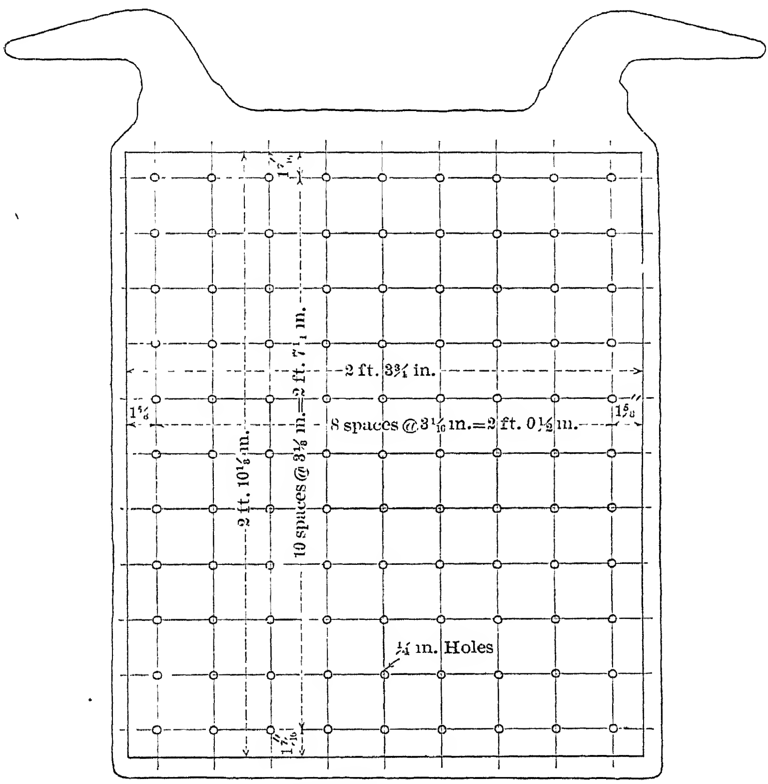


FIG. 1.—DRILLING-TEMPLATE FOR SAMPLING LARGE ANACONDA ANODES, WITH OUTLINE OF ANODE.

The anodes are carefully swept to remove foreign matter, then drilled with a 0.5-in. drill from the bottom side completely through the anode, every portion of the drillings being saved. The drillings from the entire lot are run through a grinder until the sample will pass through a 16-mesh screen. The sample is then thoroughly mixed and quartered, each quarter weighing about 1 lb. The 1-lb. sample is screened over

a 40-mesh screen, weighing the oversize and undersize, thus determining the ratio of coarse to fine, the same ratio being maintained in the assay-ton made up for analysis.

IV. Ladle-Shot Method of Sampling.

Incidentally, another method of sampling was investigated—namely, the taking of a ladleful of copper from the furnace or furnace-stream, and shotting it by pouring over a wooden paddle into water, thus making shot, which was treated in the same manner as the "regular shot sample."

V. Investigation of Smelter Shotting-Method.

1. To determine the uniformity of the charge during the pour, shot samples were taken each hour of the pour, as follows:

Lot No.	First Hour. Silver-Content. Oz. per Ton.	Second Hour Silver-Content Oz per Ton.	Third Hour. Silver-Content. Oz. per Ton.	Fourth Hour. Silver-Content. Oz. per Ton.
873	73.36	73.41	......	73.58
1000	73.79	73.79	73.82	73.79
1001	72.93	72 81	72.99	73.06
1002	76.07	75.95	76.19	76.37

2. To determine the possibility of segregation of silver in some portion of the stream as it flowed from the furnace, samples were taken from different parts of the stream. These samples showed:

	Silver-Content. Oz. per Ton.
Front of stream,	72.38
Middle of stream,	72.29
Back of stream,	72.28

3. To determine the possibility of segregation of silver by using a cold wooden paddle, shot samples were taken, using a wooden paddle and a red-hot clay paddle. These samples, on analysis, gave:

Lot No.	Wood Paddle. Silver-Content. Oz. per Ton.	Hot Clay Paddle. Silver-Content. Oz. per Ton.
936	74.50	74.60
950	73.76	73.55

VI. Investigation of Refinery Drilling-Method.

To test the uniformity of the anodes, six anodes were selected at equal intervals during the casting of a lot. Each anode was drilled in 99 places by the template; the drillings assayed as follows:

Lot No.	Anode No	Silver-Content. Oz. per Ton.
956	1	71.36
956	2	71.54
956	3	71.21
956	4	71.41
956	5	71.68
956	6	71.51

Lot No.	
885. Random anode,	73.41
885. Regular sample of lot,	73.76

Two lots of anodes were taken from lot No. 911 for drillings samples. Sample A is the regular drillings sample, obtained by drilling one hole in every fourth anode according to template, and commencing with the first anode cast. Sample B was obtained in the same manner, but starting with the third anode cast.

	Silver-Content. Oz. per Ton.
Sample A,	74.31
Sample B,	74.50

The last anode of another lot, No. 917, drilled with 16 holes, 8 on each diagonal, gave the following results:

	Silver-Content. Oz. per Ton.
Last anode,	74.52
Assay of lot,	74.76

A comparison of the results obtained by the shot-sampling method and by the drillings-sampling method on the same lots is given in Table I.:

TABLE I.—*Results of Shot-Sampling and Drillings-Sampling.*

Lot No.	Shot Sample. Silver-Content. Oz per Ton.	Drillings Sample Silver-Content. Oz per Ton.	Lot No.	Shot Sample. Silver-Content. Oz. per Ton.	Drillings Sample. Silver-Content. Oz. per Ton.
950	73.88	73.75	965	72 97	72.91
951	72.60	72.18	966	70.93	70.80
952	71.91	71.58	967	71.79	71.50
953	71.78	71.68	968	72.62	72.42
954	73.76	73.62	969	73.10	73.22
955	71.35	71.31	970	72.53	72.63
956	71.60	71.35	971	71.20	71.09
957	70.08	70.08	972	73.25	73.30
958	70.96	70.90	973	74.53	74.47
959	69.56	69.66	974	74.67	74.67
960	71.33	71.38	975	77.55	77.07
961	71.00	70.98	976	77.96	77.73
962	70.29	70.32	977	76.84	76.84
963	71.09	70.98	978	75.63	75.25
964	71.24	71.10			

The average weight of the lots given in Table I. was 106 tons.

VII. Investigation of the Ladle-Shot Method.

1. In Table II., the ladle samples were taken from the furnace, with hot and cold ladle, at the same time a regular shot sample was taken from the stream flowing out of the furnace. After pouring the shot required, the "skulls" left in the ladle were sampled by drilling.

Table II.—*Results of Using Hot Ladle and Cold Ladle.*

Lot No.	Shot Samples. Cold Ladle. Silver-Content. Oz. per Ton.	Shot Samples. Hot Ladle. Silver-Content Oz. per Ton	Furnace Shot Samples. Silver-Content. Oz. per Ton	Skulls. Cold Ladle. Silver-Content Oz. per Ton.	Skulls. Hot Ladle. Silver-Content. Oz. per Ton
967	73.83	71.74	71.35		
968	74.54	72.71	72.43		
969	73.28	73.86	73.13		
970	75.85	74.27	72.59	70.57	70.67
971	72.69	73.58	71.16	67.52	67.34
972	74.78	75.71	73.18	71.97	72.09
973	76.79	75.64	74.47	71.07	72.29
974	79.31	75.18	74.78	73.47	72.26
975	78.49	78.65	77.68	76.99	76.93
976	82.71	80.32	78.07	75.19	75.21
977	79.59	78.27	76.96	76.25	76.10
978	77.90	77.15	75.54	74.48	74.18
979	80.03	76.29	75.68	75.08	75.22
781	77.67	79.22	74.80	73.48	74.12
782	75.88	76.71	74.74	74.11	73.02
983	76.02	76.97	74.79	74.19	74.52
784	74.93	75.84	73.76	72.40	71.99
985	76.83	76.05	75.17	74.68	74.48

2. An 8-in. ladle was half filled with copper from the furnace-stream, and three separate shot samples taken from it, at the same time a shot sample was taken from the furnace-stream. The results obtained were:

	Silver-Content. Oz per Ton
Sample No. 1,	76.42
Sample No. 2,	78.67
Sample No. 3,	81.02
Skull left in ladle,	72.89
Regular shot sample taken from the furnace-stream,	74.99

3. An 8-in. ladle was entirely filled with copper from the furnace-stream, and four shot samples taken from it at the same time a shot sample was taken from the stream. The results were:

	Silver-Content. Oz. per Ton.
Sample No. 1,	75.85
Sample No. 2,	77.11
Sample No. 3,	78.40
Sample No. 4,	81.44
Skull left in ladle,	73.37
Stream shot sample,	74.99

An 8-in. ladle was half filled and allowed to cool completely. The resultant button was drilled, with the following results:

	Silver-Content. Oz. per Ton.
Top of button,	79.45
Center of button,	75.73
Bottom of button,	68.91

4. In order to see how high a result could be obtained by the ladle-shot method, an 8-in. ladle was filled with copper and allowed to stand until nearly frozen, when the metal remaining molten was shotted, with the following result:

	Silver-Content. Oz. per Ton.
Ladle-shot sample,	86.17
Correct shot sample,	73.67

VIII. Conclusion.

The conclusions reached were (1) that the shot sample taken from the furnace-stream, and the drillings of every fourth anode acording to Dr. Keller's template, will check within practical limits, and give the true silver-content of the copper, and (2) that the ladle-shot method will give high silver results, due to a segregation of silver towards the portion that solidifies last.

I wish to acknowledge the valuable assistance given by Frederick Laist, assistant superintendent, Washoe Smelter; and C. D. Demond, testing engineer, in the carrying-out of the details of this investigation.

Cyanide-Plant and Practice at the Minas del Tajo, Rosario, Sinaloa, Mexico.

BY GEORGE A. TWEEDY AND ROGER L. BEALS, ROSARIO, SINALOA, MEX.

(Pittsburg Meeting, March, 1910.)

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I. INTRODUCTION.

THE results of the cyanide-operations, given in detail in the following paper, show the progress that is being made at the Minas del Tajo. The old pan-amalgamation process, in operation up to and including 1905, while paying a profit on high-grade ores, was no longer able to treat profitably the ores in sight at that time; hence an alteration in the process was imperative. The success of this alteration is eloquently expressed by the following data:

	Pan-Amalgamation 1905.	Cyanidation. 1907.	1908
Tons treated,	51,387	58,983	69,200
Value per ton (U. S. currency),	$10.78	$8.39	$8.28
Extraction, per cent.,	65.00	88.56	87.45
Extraction, per ton,	$7.00	$7.43	$7.24
Total cost, per ton treated,	6.50	4.36	4.01
Profit, per ton treated,	0.50	3.07	3.23

The item "total cost" in the foregoing tabulation includes everything—working-costs, development, management, general expenses, bullion-expenses, and taxes.

II. Definitions and Explanations.

The following definitions and explanations are given for the sake of clearness:

Slime.—That product, clayey in nature, consisting largely of decomposed feldspars and having the characteristics of a colloid.

Fine Sand.—Sand finer than 150-mesh, having a gritty feel and which will settle in water, leaving it clear in less than 1 minute.

Medium Sand.—Sand in size between 40- and 150-mesh.

Coarse Sand.—Sand in size coarser than 40-mesh.

Residue.—When used in speaking of assays, this term refers to samples dried without washing; in other words, containing a certain amount of cyanide solution and dissolved values.

Undissolved.—Assay of pulp after thorough washing (that is, after removal of all cyanide solution and dissolved values).

Ton.—The short ton, 2,000 lb., is used in all instances.

Cyanide.—All strengths and figures are in terms of potassium cyanide.

Sodium Cyanide, equivalent to 130 per cent. KCN, is used in the plant.

Screen-Tests are made with the following sieves:

Mesh	40	60	80	100	150
Openings, mm	0.0407	0.0216	0.0169	0.0127	0.0083

Assay-Values and Costs are given in United States currency.

III. Ore.

Two classes of ore are mined and treated—namely, oxidized and unoxidized ore. Roughly speaking, all ore coming from below the 150-ft. level falls into the latter class. Of these two classes there are several types, differentiated by the ratio of their silver- and gold-content, proportion of heavy sulphides, and amenability to treatment by cyanidation. Each type in the unoxidized ore has its counterpart in the oxidized ore.

1. *Unoxidized Ore.*—Type 1, characterized by large propor-

tion of galena, not rich in silver, and presenting no difficulty in treatment.

Type 2, carrying a lesser proportion of galena, accompanied, however, with zinc-blende and chalcopyrite. Whenever this chalcopyrite is found the ratio of gold to silver increases. This type of ore is amenable to treatment, but causes a larger cyanide-consumption.

Type 3, clean ore, carrying very little base sulphides, with a greater ratio of silver to gold, and easily cyanided. The gangue of this type contains a considerable amount of rhodochrosite and feldspar.

2. *Oxidized Ore.*—The counterparts of these types in the oxidized ore are traced by their silver-gold content ratio, and by the presence of analogous minerals derived from the oxidation of the base sulphides, etc. For instance, Type 1 shows considerable lead carbonate; Type 2, a lesser amount of lead carbonate, accompanied by zinc carbonate and copper carbonates and oxides, while Type 3 contains manganese oxides, particularly psilomelane or wad.

All these types are amenable to cyanide-treatment, Type 2 causing the greatest cyanide-consumption, while Type 3 produces the most slime, due to the greater proportion of clay formed by the decomposition of the feldspars.

A typical analysis of Type 3, oxidized ore, is:

	Per Cent.
Copper,	0.2
Iron,	4.2
Zinc,	1.4
Manganese,	1.6
Lead,	trace
Lime (CaO),	1.2
Sulphur,	1.5
Insoluble (Al_2O_3–SiO_2),	82.8
Undetermined,	7.1
	Oz Per Ton.
Gold,	0.20
Silver,	8.60

The sulphur is partly combined as sulphide and partly as the sulphates, $FeSO_4$ and $Fe_2(SO_4)_3$, the acid sulphate accounting for the greater acidity of this class of ore. The undetermined comprises CO_2, oxygen of the sulphates, and oxygen combined with manganese, etc.

IV. Preliminary Experiments.

Before the erection of the mill and cyanide-plant experiments were conducted to determine the best method of treatment. A comparison of the results obtained experimentally with the results in the completed mill is interesting.

The ore experimented upon was the unoxidized class. Samples were taken from various parts of the mine and treated separately, then a general sample of the unoxidized ore was tested. The first work was with 100-lb. samples, these being crushed and ground in a sample-grinder and classified into sand and slime. The sand was leached in small tubs, 18 in. in diameter by 18 in. deep, while the slime was treated by agitation in the same sized tubs fitted with cross-arm stirrers.

Extremely erratic results were obtained with the sand-leaching. The lower half of the sand in the tubs behaved as does the lowest 6-in. layer in a large tank, being slow to drain and retaining a great deal more moisture than the top sand; the upper half behaved as does the top 6-in. layer of sand in large tanks. The varying amount of solution retained by the sand destroyed the value of the work, and it is for this reason that leaching-tests in small tubs were unreliable. On the other hand, the tubs fitted with stirrers for treating the slime more nearly approximated the working-conditions, the results being uniform and reliable.

The dissolution of the gold and silver and the greatest permissible size of grain were determined by bottle-tests, using bottles holding from four to seven times the volume of air to that of the pulp; these tests showed that on ore finer than 40-mesh more than 90 per cent. of the values could be dissolved in four days.

Tests for the determination of the acidity of the ore were made in bottles, agitating the samples with solution of sodium hydrate; the average of the tests gave an acidity equal to 7 lb. of lime per ton of ore, this high consumption being due chiefly to the ore from old stope-filling containing organic matter.

The lime obtainable contained only 40 per cent. of effective CaO; thus the actual lime-consumption was from 14 to 17 lb. per ton of ore.

Bottle-tests for cyanide-consumption indicated a chemical consumption of from 1.5 to 2.1 lb. per ton, provided the pro-

tective alkalinity did not drop below 0.02 per cent. of CaO. This was corroborated by tests of sand in 5-ton lots. It was found that, even after 10 days' leaching, the cyanide-consumption would rapidly rise if the protective alkalinity dropped below 0.02 per cent. of CaO. Lime crushed not finer than 20-mesh gave the best results; this, mixed with the ore, slowly went into solution as needed, the leachings coming off alkaline to the end of the treatment.

Milk of lime was unsatisfactory, as it gave up its alkalinity too quickly.

V. Mill-Run Tests.

The feasibility of collection of sand, leachability, and extraction on a working-scale were determined by small mill-runs, the ore being crushed in 5- and 10-ton lots in a five-stamp battery; the pulp was run to a pit, where the sand was collected, the slime overflowing through a gate and settling in a tank below the pit, the overflow being regulated by raising the gate.

In the mill-runs a punched screen having openings equal to 0.6 mm. was used, and screen-tests showed approximately 35 per cent. slime, while from 3 to 8 per cent. of the sand was coarse.

The value and amount of the various sized products are given in Table III., which shows that the chief values lie in the medium to fine sand, while the product coarser than 40-mesh is lower in value than the original sample.

On this latter product an extraction of about 60 per cent. was obtained. Allowing 10 per cent. of this size in the sand separated from ore carrying 8 oz. of silver per ton, there would be, on a 1-ton basis, 200 lb. of pulp carrying from 4 to 6 oz. per ton, of which 60 per cent. could be extracted, leaving 200 lb. of pulp, assaying 2 oz. of silver per ton, and representing 0.2 oz. of silver per original ton of ore. By regrinding this product a further extraction representing 0.1 oz. of silver per ton of ore could be recovered. This extra recovery from fine grinding would not pay for the cost of the work.

The sand collected was found to be easily leachable, though containing 7 per cent. of slime. Charges of 5 tons were treated from 8 to 10 days. The strength of solution found best in the bottle-tests—namely, 0.3 per cent. of KCN—was used for the first two days' treatment. The tank was then aërated for a

TABLE I.—*Results of Leaching-Tests on Sand.*

Weight of sand, 3.37 tons.
Per cent. of moisture, 18.
Weight of lime added, 32 lb.
Charging-value of sand, 0.42 oz. gold, 10.0 oz. silver per ton.

Screen-Test.		Gold-Assay.		Silver-Assay.	
Size	Quantity.	Heads.	Tails	Heads.	Tails
	Per Cent.	Ounces Per Ton	Ounces Per Ton.	Ounces Per Ton.	Ounces. Per Ton
On 40............	3.9	0.25	0.040	6.2	2.4
On 60...........	20.0	0.28	0.022	5.3	1.2
On 80............	21.7	0.30	0.015	7.2	0.9
On 100...........	5.8	0.29	0.010	7.4	1.0
On 150...........	28.1	0.50	0.014	10.0	1.1
Through 150....	20.5	0.71	0.012	19.0	1.4

Solution.	Strong Solution.		Leachings		Consumption KCN Per Ton Solution.	Undissolved.		Residue.	
	KCN.	CaO.	KCN.	CaO.		Au.	Ag.	Au.	Ag.
Tons.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Pounds.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton	Oz Per Ton.
2.31	0.34	0.12	0.24	0.10	4.62	0.065	3.32	0.28	8.0
2.13	0.22	0.10	0.22	0.15	......	0.03	2.07	0.12	6.0
2.07	0.27	0.09	0.28	0.06	......	0.01	1.74	0.04	3.5
1.78	0.30	0.07	0.28	0.06	0.71				
1.61	0.28	0.09	0.26	0.08	0.64	0.01	1.45	0.03	3.0
1.28	0.27	0.09	0.24	0 09	0.76	0.012	1.29	0.02	2.2
1.28	0.26	0.09	0.24	0 10	0.76	0.01	1.22	0.016	1.6
1.03	0.54	0.12	0.26	0.08	5.76	0.011	1.16	0.014	1.3
0.55	0.36	0.12	0.30	0.08	0.66				
0.26	water	wash	0.38	0.07	+2.17				
2.00	water	wash	0.10	0.06	+4.35	0.012	1.15	0.015	1.17

Percentage of extraction by residue, gold, 97.0; silver, 89.6 per cent.
Cyanide-consumption, 2.2 lb. per ton of sand.
Time of treatment, 11 days. Solution added twice daily.

day, and solutions again added. Between the additions of solution the tank was thoroughly drained. The leachings were passed through a zinc-box and returned to the storage-tank, the precipitated solution thus being used repeatedly.

The effectiveness of the solution after use was determined by using 2 tons of solution for the treatment of five lots of ore of 5 tons each. The solution was in use for six weeks, being precipitated after each passage through the sand, and evaporation-losses made up by addition of water. At the end of this period it was as effective for all purposes as at the beginning. The precipitation was good, the solution leaving the boxes

carrying 0.05 oz. of silver per ton, and a trace of gold. The consumption of zinc was 0.8 lb. per ton of ore treated.

The net result of these leaching-tests showed:

1. That the extraction of gold and silver was good.

2. That the dissolution of the gold was very rapid, the silver going into solution more slowly.

3. That the best results were obtained by allowing the charges to drain thoroughly before each addition of solution.

4. That the leaching-rate was 4 to 6 in. per hour, and that no serious fouling of working solution took place.

Full data of one of the leaching-tests are given in Table I.

The collected slime from the test-runs was treated, in small tanks fitted with mechanical stirrers, by agitation and decantation in solution averaging 0.20 per cent. of KCN. The extraction was: gold, 94, and silver, 88 per cent.

The consistency of the pulp giving the best results was 4 of solution and 1 of dry slime. The slime settled to 52 per cent. of moisture in 12 hours.

The dissolution of the silver, commercially profitable to dissolve, took place in the time necessary to mix and decant the solution, giving five decantations, hence the washing-out of the values dissolved was the most important part of the test.

Table II. gives the results of the treatment of one charge of slime.

TABLE II.—*Results of Tests on Slime.*

Weight of wet slime charged, 20 kg.
Assay, gold, 0.20 oz. per ton; silver, 9.60 oz. per ton.
Moisture in charge, 45 per cent.
Weight of dry slime, 11 kg.
There was added at the beginning of the first treatment 120 g. of KCN and 105 g. of lime.

Water Added.	Time of Treatment.		Solution Decanted.			Undissolved.		Residue.	
			Quantity.	Composition.		Gold	Silver.	Gold.	Silver.
				KCN.	CaO.				
Kg.	From	to	Kg	Per Cent.	Per Cent.	Oz. Per Ton	Oz Per Ton.	Oz. Per Ton.	Oz Per Ton.
35.0	8 a.m.	5 p m.	27.0	0.23	0.05				
......	....	2 p.m.	......	......	......	0.02	3.80		
......	....	5 p.m.	......	......	......	0.01	1.29		
30.0	2 p.m.	6 p.m.	29.0	0.09	0.02	0.007	0.79	0.015	1.49
30.0	9 a.m.	2 p.m.	29.0	0.05	0.01	....	......	0.007	0.88

Percentage of extraction by residue, gold, 95; silver, 90 per cent.
Cyanide-consumption: Chemical, 1.6; mechanical, 1.7 lb. KCN; total, 3.3 lb. of KCN per ton.

Table III. gives the results of one of the battery-runs, showing the amount and assay of the various-sized products of the battery-pulp before treatment, and of the classified sand before and after treatment.

TABLE III.—*Results of the Battery-Tests.*

Screen Size.	Battery-Sample			Sand-Sample				
	On Screen	Value		On Screen	Heads.		Tails	
		Gold.	Silver		Gold.	Silver.	Gold.	Silver
	Per Cent.	Oz per Ton.	Oz per Ton.	Per Cent.	Oz per Ton.	Oz per Ton.	Oz per Ton.	Oz. per Ton
On 40..................	2.51	0.24	6.1	3.9	0.30	6.2	....	2.5
On 60..................	11.08	0.19	4.2	20 0	0.29	5.4	....	1.6
On 80......	1.52	0.25	5.8	21.7	0.30	7.2	....	1.2
On 100................	3.70	0.27	5.8	5.8	0.29	7.4	....	0.83
On 150........	17.30	0.42	8.6	28.1	0.50	11.4	....	1.2
Through 150..........	50 1	0.61	14.1	20.5	0.86	20.2	....	1.5
Through 150, sand..	16.5	1.10	19.2	14.8	1.05	24.0		
Through 150, slime.	33.6	0.37	11.7	5 7	0.19	10.0		

VI. CONCENTRATION EXPERIMENTS.

To determine the advisability of concentration of the ore, a sample of the sand in each charge was concentrated before and after treatment with cyanide. The amount and value of the concentrate obtained were not sufficient to justify concentration; moreover, the concentrate obtained from the treated sand showed that a good extraction was obtained from it in the tanks. A slightly-increased extraction could be made by separating and regrinding, but this increased extraction was not sufficient to justify the erection of an extra plant.

The average results of a number of tests were:

		Assay-Value. Gold. Oz Per Ton.	Silver. Oz. Per Ton.
Concentrate from sand before treatment, .		3.25	26.7
Amount of concentrate recovered, . .	1.2 per cent.		
Ounces in concentrate per ton of sand, .		0.039	0.3
Concentrate after treatment of sand, . .		0.62	4.6
Ounces in concentrate per ton treated sand,		0.006	0.05
Extraction on concentrate by cyanidation of sand, ...		79.9	82.8

VII. OUTLINE OF OPERATIONS.

The ore from the mine is crushed by a No. 5 Gates breaker, style K, to pass a 2-in. ring. The breaker is fed from two 200-

ton bins, and the ore, after crushing, is elevated by inclined belt to the fine-ore bins.

The mill and cyanide-plant are situated on the Rosario river, half a mile from the mine, the ore being hauled to the mill by a light locomotive in gable-bottom cars, holding 15 tons. A

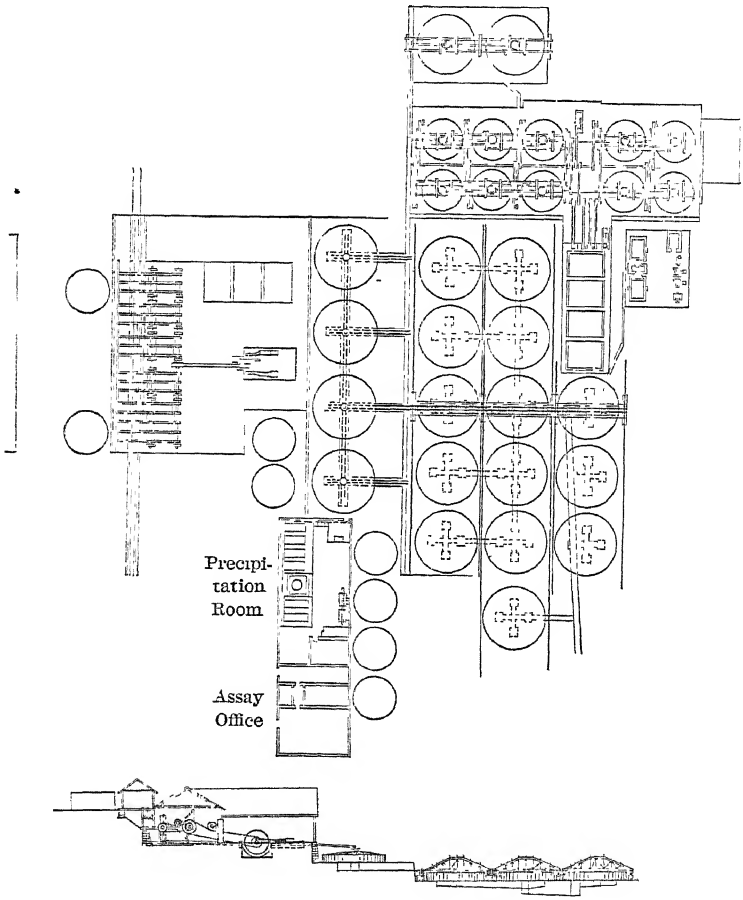

FIG. 1.—PLAN AND SECTIONAL ELEVATION OF MILL AND CYANIDE-PLANT.

plan and elevation of the mill and cyanide-plant are given in Fig. 1, and various photographic views of the plant in Figs. 2 to 5 and Figs. 7 and 15. The bins at the mill have a capacity of 600 tons.

The ore is fed from the mill-bins by automatic feeders to 12 batteries of 5 stamps each, crushing in cyanide solution.

Lime is added to the ore in the bin as needed. The alkalinity of the battery-solution is maintained at 0.05 per cent. of CaO.

From the batteries the pulp flows directly to four sand-collecting tanks, each holding 145 tons of dry sand. The pulp is distributed by radial-arm distributors, fitted with ball-bearings, each tank being equipped with a distributor. The sand settles, and the slime overflows through four curtained gates to a launder leading to the slime-plant.

The two slime-collecting tanks are 35 ft. in diameter by 9.5 ft. deep, set in series, so that the pulp enters one and overflows to the second. The overflow from the second tank runs to a concrete sump, and is then pumped back to the battery-supply tank above the mill.

The sand, after collection and thorough draining, is transferred by cars to the leaching-tanks, 14 in number. After treatment the sand is sluiced to the river.

The thickened slime is withdrawn at regular intervals from the slime-collecting tanks. The treatment of this slime, by agitation and settlement, takes place in nine tanks, each 24 ft. in diameter by 9.5 ft. deep. From the treatment-tanks the slime is transferred to a tank of the same size, supplying the filters. After filtration the slime is washed to the river.

The solutions from the sand- and slime-plants are collected in sumps built of brick and faced with cement. Centrifugal pumps, driven by the engine running the slime-plant, handle the solutions. The water for the mill and mine is supplied by steam-pumps situated in a well connected by a tunnel with the river.

VIII. Milling.

The mill consists of 12 batteries of 5 stamps set on concrete-mortar blocks. The battery-frame is of the front-knee pattern, the sills anchored in concrete by 1.5-in. bolts. In laying the concrete great care was taken in tamping and in adding but small amounts of material at a time. The blocks have not shown the slightest tendency to crack.

Each mortar weighs 5 tons. Though designed for double discharge, they now have the backs blocked up with a heavy cast-iron plate and are fitted with liners. The mortars were planed on their bases, and the tops of the concrete blocks were finished carefully with neat cement. Between mortar and block

a $\frac{1}{8}$-in. sheet of rubber is placed; eight bolts, 1.5 in. in diameter, are used to hold the mortar rigidly in place on the block.

The gross weight of the falling-stamp with new shoe is: stem, 450; boss, 350; tappet, 120; shoe, 180; total, 1,100 lb. The average running weight is 1,010 lb. The weight of the die used, when new, is 140 pounds.

The stamps drop 6.5 in. 104 times per minute. Wooden guides are used, which last from eight to ten months; the wood is native and is extremely hard and of fine grain.

A cross-compound, slide-valve engine drives a line-shaft running the batteries, eight having separate drive, while four are driven in pairs, ten stamps to a shaft; however, the separate drive for each battery is preferable. Idler pulleys are used as belt-tighteners. For the 60 stamps 125 h-p. is required.

The cam-shafts driving five stamps are 6 in. in diameter, while those driving ten stamps are 6.5 in. The cams are of the Blanton self-locking type.

When crushing with double discharge it was found that an excessive amount of solution was necessary in order to maintain a proper wash in the mortars, from 10 to 12 tons being required per ton of ore. With single discharge, the same amount was crushed with 6 tons of solution. It was for this reason that the back discharge was blocked up with a cast-iron plate and the mortar narrowed by liners. Chuck-blocks are used to maintain the discharge at from 2.5 to 3 in., which is the height of greatest capacity.

The endeavor was to obtain as great a capacity per stamp as possible, with the condition of keeping the amount of coarse sand under 6 per cent. Various types of screens have been tried to achieve this result.

Using woven-wire screens, crushing in cyanide solution carrying from 0.04 to 0.08 per cent. of CaO, the lime built up on the screens and choked them, closing the holes as with a cement. Frequent brushing was necessary to keep the holes clear, and this broke the screens; 30-mesh iron-wire screens were tried and discarded. Heavy punched plate having holes 0.6 mm. in diameter gave a proper-sized product, and no trouble from lime-deposition; but the capacity was low, since the holes were spaced too far apart and the plate was too heavy. Diagonal-slot screens, the slots 0.6 mm. wide, gave capacity, but as they

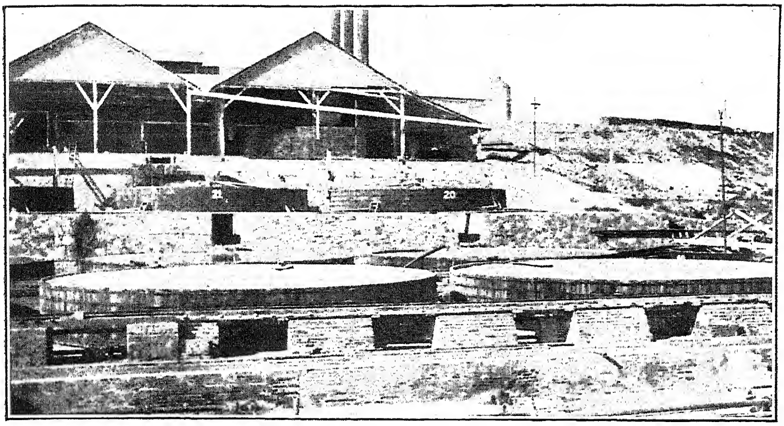

FIG. 2.—VIEW OF PLANT; SAND-TREATMENT TANKS IN THE FOREGROUND.

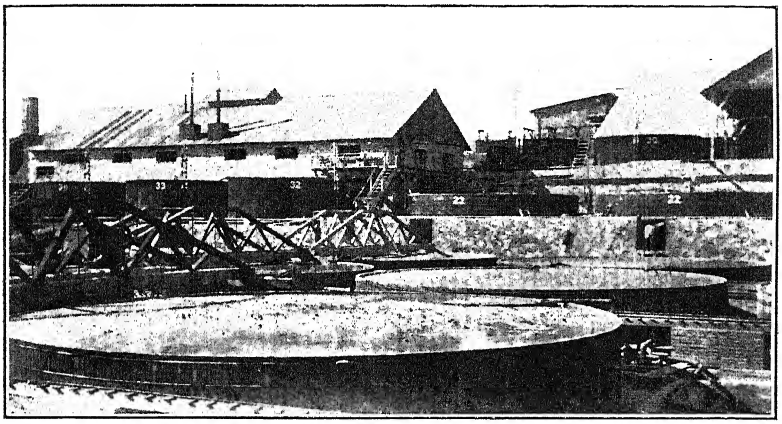

FIG. 3.—VIEW OF PLANT; SAND-TREATMENT TANKS IN THE FOREGROUND.

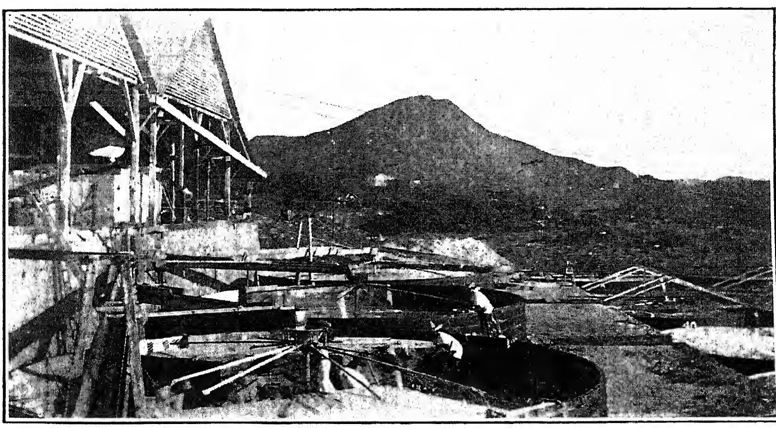

FIG. 4.—COLLECTION-TANKS WITH RADIAL-ARM DISTRIBUTORS.

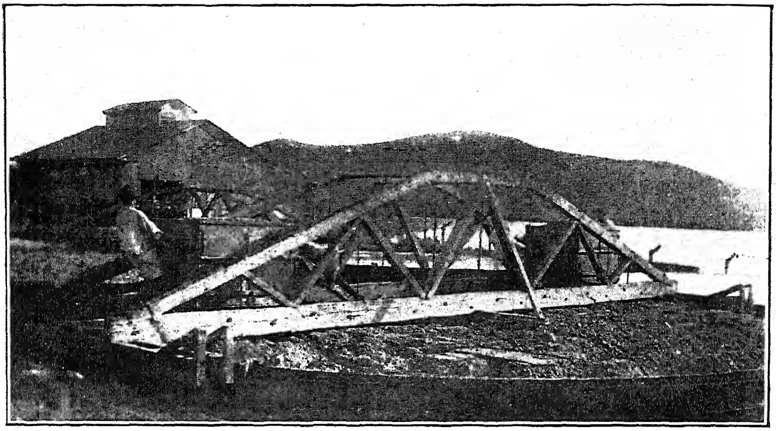

FIG. 5.—LEACHING-TANKS, TRAVELING-BRIDGE FOR CHARGING SANDS.

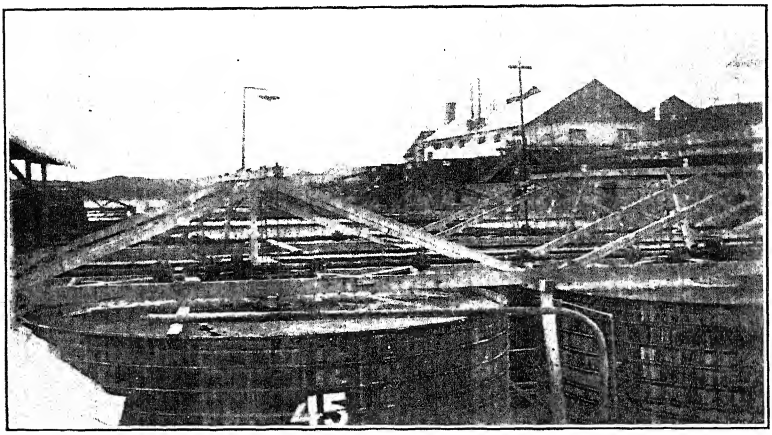

FIG. 7.—SLIME-TREATMENT TANKS WITH OVERHEAD TRESTLES.

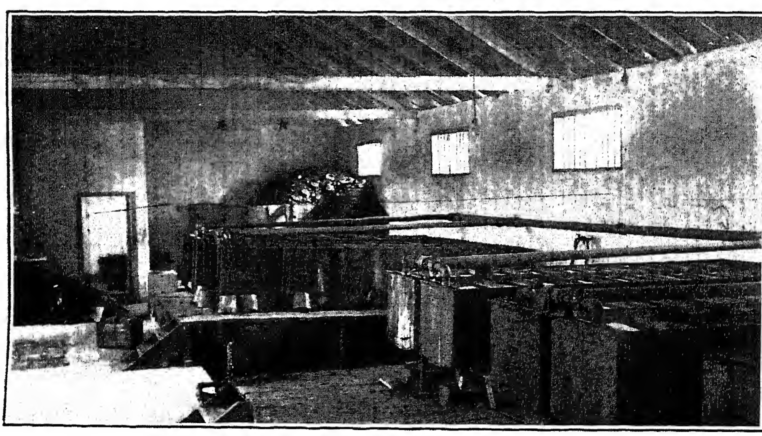

FIG. 15.—PRECIPITATION-ROOM, SHOWING THE ZINC-BOXES.

rapidly wore to 1 mm. and larger at the lower end of the slot, the product was too coarse.

It was finally found that thin punched tinned screens gave the best results; these last 15 or 20 days, as compared with 30 or 40 days when using the heavy plate. Their cost is such, however, that though lasting but half the time, the screen-cost per ton crushed is one-third of that resulting from the use of heavy plate.

The stamp-duty varies with the class of ore. The oxidized ore coming from the open-cut and upper levels of the mine is very soft and friable, consisting largely of decomposed andesite with the feldspars changed to clay. Unoxidized ore from the lower levels is hard; much of the quartz in it is like flint. Crushing this latter, the stamp-duty is about 3.5 tons. With a mixture of the two classes containing from 20 to 30 per cent. of oxidized ore, the stamp-duty is increased to 4 tons.

The quantity of slime in the pulp from the stamps increases from 35 per cent. in crushing unoxidized ore to 45 per cent. in crushing a mixture of oxidized and unoxidized, the screens, discharge, drop, and solution used being the same in each case.

The number of tons crushed and the quantity of slime per stamp were:

Screen Used	Tons Per Stamp.	Slime. Per Cent.
0 6 mm., heavy plate.	3.25	35.5
0.6 mm., heavy plate.	3.60	40.4
1.0 mm., heavy plate.	4.00	41.4
0.6 mm., thin tinned plate.	4.04	44.5
0.6 mm., thin tinned plate.	4.18	45.2

The character of the product of the stamps when using different screens is given in Table IV.

TABLE IV.—*Results of Using Various Screens.*

Sample of Pulp.	On 40.	On 60	On 80.	On 100.	On 150.	Through 150.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.
A.	12.0	18.0	15.0	10.0	10.0	35.0 (5 of sand, 30 of slime).
B.	8.0	18.0	16.0	11.0	11.0	36.0 (7 of sand, 29 of slime).
C.	5.0	14.0	12.0	9.0	18.0	42.0 (10 of sand, 32 of slime).
D.	4.5	16.5	15.5	15.0	13.5	35.0 (9 of sand, 26 of slime).

A. Through a 0.6-mm. slot screen. C. Through a 0.6-mm. punched plate.
B. Through a 1.0-mm. punched plate. D. Through a 0.6-mm. tinned plate screen.

Different makes of shoes and dies have been tried, but no advantage was found in using a shoe of one class and a die of another. Good steel shoes and dies give no trouble from cupping; the shoe wears down from 180 to 40 lb. and the die from 140 to 35 lb. Cast-iron dies were tried and found unsatisfactory, costing more per ton crushed, and wearing unevenly; moreover, they did not keep the shoe in any better condition than the steel dies. The consumption of steel in shoes, per ton of ore crushed, was 0.52 pound.

The pulp from the batteries is delivered to a V-shaped launder running to the sand-collecting tanks. The launder is set on a grade of 2.75 per cent., which is sufficient to keep the launder clear, when using 6 tons of solution per ton of ore. A straight launder would carry the pulp with less grade, but where the pulp from the stamps enters, the flow in the launder is rhythmical, and a lesser grade was found insufficient to carry the ore.

As the mill-site is comparatively flat, launders were placed at the minimum possible grade; in fact, it was only by doing this that the necessity of elevating the pulp was avoided.

Table V. shows the average costs of milling 41,465 tons of ore, covering a period of seven months:

Table V.—*Summary of Costs of Milling.*

	Per Ton.
Labor, native,	$0.0583
Employees, white,	0.0080
Shoes,	0.0275
Dies,	0.0162
Belting, lacing,	0.0116
Stems, screens,	0.0070
Cocks, valves,	0.0021
Lubricant and miscellaneous,	0.0075
Power,	0.3500
Total,	$0.4882

The labor-cost is total, including that in the shops and mill indirectly chargeable to crushing, as also maintenance and repairs. The power-item is 71 per cent. of the cost of milling. Wood is used as fuel and costs $5.50 per cord of 180 cubic feet.

IX. Cyaniding.

1. *Sand-Plant.*

There are four redwood collecting-tanks, each 35 ft. in diameter by 4.5 ft. deep, fitted with pine filter-bottoms supporting cocoa-matting and 6-oz. duck filter-cloth. The filter-cloth is protected by 1.5-in. strips spaced 2 in. apart. The filter and strips take up 6 in. of the depth of the tank.

Each tank has a radial-arm distributor, and four side-outlets for the overflow of slime. The outlets are 4-in. pipes, and have a lattice and curtain in front to control the overflow. There are also four bottom-doors.

The pulp from the batteries is delivered to the hopper of the distributor and spread over the tank (Fig. 4). The sand settles, while the slime overflows the curtains and is delivered to a launder running to the slime-plant. As the sand collects the curtains are raised, the endeavor being to overflow as much of the slime as possible. During the collection and until the charge is transferred to the treatment-tanks, the sand is drained by two pipes leading from the bottom of the tank. A charge amounting to 145 tons drains to 15 per cent. of moisture after 30 hours.

Charges are allowed to stand as long as possible before transferral, usually 60 hr. The sand is then shoveled through the bottom discharge-doors to cars, and charged into the treatment-tanks. Low, wide, gable-bottom 2-ton cars are used. The turn-tables beneath the tanks have ball-bearings which permit very easy handling of the cars.

The cars containing the sand are trammed over the treatment-tanks on movable bridges (Fig. 5). Two cars are handled at a time; the sand is charged evenly over the surface of the tank and leveled off. The traveling-bridges are a cheap and efficient means of filling the tanks, and no vibration is transmitted to the tank to cause settling and channeling of the sand. The work of charging, done by contract, costs $0.05 per ton.

The leaching-tanks, 14 in number, are the same size as the collecting-tanks. The canvas filter is held in place by 0.5 by 1 in. strips spaced 5 in. apart and radiating from the discharge-

doors, which protect the cloth while the sand is being discharged.

In charging the tanks the sand increases 20 per cent. in volume. Strong solution is run on immediately after charging, and the sand allowed to soak for 6 hr. The charge subsides during this time and occupies about 24 cu. ft. to the ton. Before charging, the sand will occupy from 20 to 22 cu. ft. per ton.

The sand leaches rapidly and drains to 25 per cent. of moisture in 6 hr. After the first addition of solution and draining, the leaching-cocks are left open for all succeeding leaches till the tank is ready to be discharged.

The solution from the slime-plant, consisting of the decantations from the last treatments of the slime, is pumped to two supply-tanks situated above the sand-plant. This solution, which carries from 1 to 1.5 oz. of silver and from 0.03 to 0.06 oz. of gold, is brought up to a strength of 0.25 per cent. of KCN by addition of cyanide. This solution is used for the first or strong washes on the sand, being run on the tanks every 8 hr., with continuous draining. The tanks drain to from 22 to 25 per cent. of moisture between the additions of solution.

When the silver dissolves rapidly the addition of strong solution is stopped after three days, and solution from the slime-plant is used without addition of cyanide. It takes from three to seven days to bring the undissolved value of the sand to less than 2 oz. of silver per ton. The charge is then allowed to aërate from two to three days.

The solutions from this first period of treatment are collected in a sump and pumped to the high-grade solution-tank above the precipitation-boxes.

After precipitation, the solution is used for the barren leaches, being put on the sand after it has been thoroughly aërated. This precipitated solution, titrating from 0.15 to 0.25 per cent. of KCN, is used for from three to six days until the residue and undissolved values of the sand are approximately equal. When the leaches drop in value to 0.4 oz. of silver and 0.01 oz. of gold per ton, the tank is allowed to aërate a second time for two or three days. The leachings during this second period are collected in a separate sump. The sump-solution, assaying from

1 to 2 oz. of silver per ton, is pumped to the second tank above the precipitation-boxes.

After the second aëration, water-washes, in lots of 10 tons each, are continued usually for two days, and the drainings drop to 0.1 oz. of silver per ton, a trace of gold, and a cyanide-content of from 0.02 to 0.03 per cent. of KCN.

After the final water-washes, the tank is drained and sluiced to the river through the four bottom-doors. It takes from 8 to 10 hr. to discharge the sand, since the tanks are shallow. One man using a 2-in. hose, under a 40-ft. head, washes out the sand, the launders carrying the discharged sand having a 3-per cent. grade.

The total time of treatment is approximately 18 days, which allows for from four to six days' aëration in the tanks. Without this aëration a much larger quantity of solution is necessary to accomplish the work.

The leaching-tanks are fitted so that the precipitated solution or barren washes may be applied from beneath. This is seldom done, however, except in the case of a charge which is slow to dissolve.

The work in the plant has shown that, provided the metal in the ore is soluble, it can be dissolved and removed from sand as rapidly as from slime, and more cheaply. The essential point is that the charge of sand shall be uniform and leachable. The cost of treating sand is much less than that for slime, and for this reason the endeavor has been to treat as sand as much of the product as possible.

The ore treated during the first year of the present plant was almost entirely unoxidized ore, and weak solutions were used. The second year a large amount of oxidized ore, often containing much organic matter from the surface-soil mixed with it, was treated. This ore contained in addition a large amount of manganese dioxide in the form of wad, and gave difficulty in both the sand- and slime-plants. To date, no certain remedy has been found to overcome the difficulty due to the manganese. The trouble is one that will have to be solved by chemistry. Fine-grinding tests have shown no increase in extraction. The use of bromocyanogen has, in some experiments, given a slightly better extraction than with cyanide alone.

Tables VI. to XI. give the details of operations at the sand-plant.

TABLE VI.—*Results of Operations at the Sand-Plant*, 1907.

The ore treated was unoxidized mine-ore. The strength of the strong solution was 0.20 per cent. of KCN.

1907, Month	Quantity of Sand	Assay of Heads		Values in Sand.		Assay of Tailings.		Quantity Recovered.	
		Gold	Silver	Gold.	Silver	Gold	Silver	Gold	Silver
	Tons.	Oz per Ton.	Oz per Ton.	Fine Oz.	Fine Oz	Oz per Ton.	Oz. per Ton.	Fine Oz.	Fine Oz
Mar.	3,190	0.168	4 68	536 50	14,934 0	0 01	0 77	504 60	12,469.0
April	2,900	0 147	5 09	429.20	14,762 0	0.01	0 75	400.20	12,572 5
May. ..	3,190	0.139	4 78	443 70	15,257.5	0.01	0 73	411.80	12,908 5
June ...	3,190	0.167	4 63	533.30	15,094.5	0.01	0.78	501 40	12,586 0
July .	3,335	0.149	3 45	497.35	11,513.0	0.01	0.74	474.00	9,033 5
Aug.. . ..	3.080	0.188	4 97	579.60	15,330 0	0 01	1.24	548 80	11,508 0
Total....	18,885	0 159	4.60	3,019 65	86,891 0	0 01	0 84	2,840 80	71,077 5
		$5.59		$105,861 67		$0.63		$94,258 08	

Average recovery for the six months: gold, 94.0; silver, 81.8 per cent. With silver at $0.50 per oz. the combined extraction is 89.0 per cent.

TABLE VII.—*Results of Operations at the Sand-Plant*, 1908.

There was an increased amount of oxidized ore in July, while the ore milled during most of the period was nearly half oxidized. The strength of the strong solution was 0.20 per cent. of KCN.

1908. Month.	Quantity of Sand.	Assay of Heads.		Values in Sand.		Assay of Tailings.		Quantity Recovered.	
		Gold.	Silver.	Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	Tons.	Oz per Ton	Oz. per Ton.	Fine Oz.	Fine Oz.	Oz. per Ton.	Oz per Ton.	Ounces.	Ounces.
Feb...	3,045	0.181	4.52	553.90	13,775.0	0.01	0.94	523.45	10,904.0
Mar'.... ..	3,335	0.157	4.82	524.90	16,080.5	0.01	0.97	491 55	12,832.5
April......	3,480	0.137	4 96	482.85	17,284.0	0.01	0.49	438.05	15,585.3
May. . .	2,900	0.156	5.33	453.85	16,457.0	0.01	1.10	424.85	13,267.0
June. ..	3,045	0.171	5.73	521.00	17,458.0	0.02	1.10	460.10	14,108.5
July.. . ..	3,045	0.180	5.59	626 40	19,474.0	0.02	1.50	556.80	14,254.0
Total ..	18,850	0 167	5.33	3,162 90	100,528.5	0.013	1 03	2,894.80	80,951.3
		$6.12		$115,641.39		$0.78		$100,311.16	

Average recovery for the six months: gold, 91.5; silver, 80.5 per cent. With silver at $0.50 per oz. the combined extraction is 86.5 per cent.

TABLE VIII.—*Results of Operations at the Sand-Plant*, 1909.

1909, Month.	Quantity of Sand	Assay of Heads.		Assay of Tailings.		Recovery.	
		Gold	Silver.	Gold	Silver.	Gold	Silver.
	Tons.	Oz per Ton	Oz. per Ton.	Oz. per Ton.	Oz. per Ton	Per Cent.	Per Cent.
March	3,190	0.269	5.69	0.02	1.15	92.5	79.8
April	3,045	0.241	6.25	0.02	1.40	91.7	77.6
May	3,190	0 260	6.55	0.02	1.30	92.3	80.15
June	3,335	0.231	6.52	0.02	1.30	91.3	80.0
	12,760	0.250	6.25	0.02	1.29	92.0	79.3
		$8.28		$1.05		87.3	

A diagrammatic chart showing the dissolution and extraction of gold and silver from a charge of sand is given in Fig. 6.

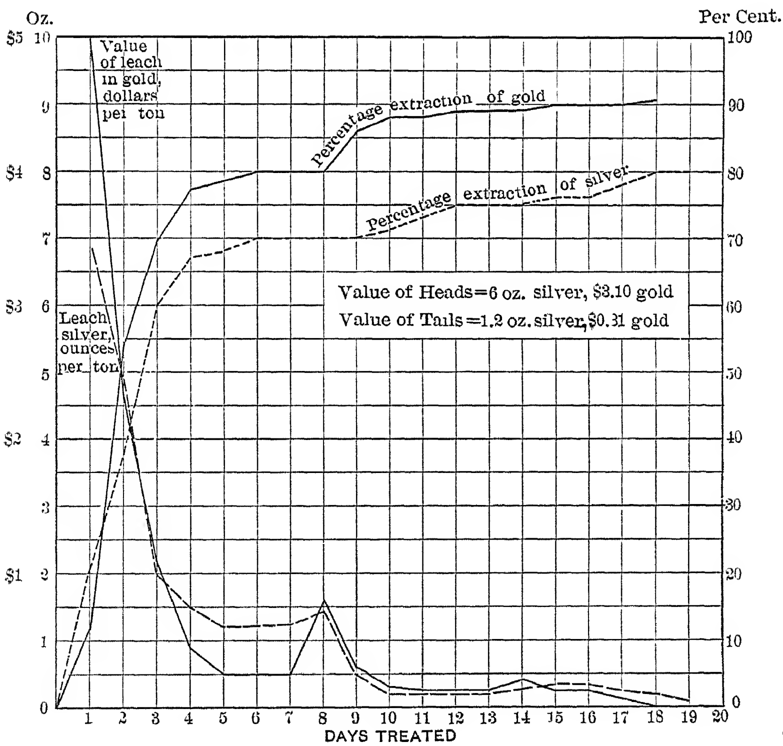

FIG. 6.—CHART SHOWING DISSOLUTION AND EXTRACTION OF GOLD AND SILVER FROM A CHARGE OF SAND.

TABLE IX.—*Screening-Tests at the Sand-Plant.*

Size	Unoxidized Ore: Quantity	Unoxidized Ore: Heads Gold	Unoxidized Ore: Heads Silver	Unoxidized Ore: Tails Gold.	Unoxidized Ore: Tails Silver.	Oxidized Ore: Quantity	Oxidized Ore: Heads Gold	Oxidized Ore: Heads Silver	Oxidized Ore: Tails Gold.	Oxidized Ore: Tails Silver.
	Per Cent.	Oz per Ton	Oz per Ton	Oz per Ton	Oz per Ton	Per Cent	Oz per Ton.	Oz per Ton.	Oz per Ton.	Oz. per Ton.
Unscreened	......	0.34	4.5	0.015	1.0	......	0.22	6.4	0.02	2.1
Screen 40	9.0	0.12	2.2	0.025	1.4	11.0	0.20	4.4	0.02	1.8
Screen 60 ...	20.0	0.16	2.6	0.020	0.9	24.5	0.14	4.0	0.02	2.0
Screen 80	22.0	0.40	2.8	0.012	0.8	20.5	0.14	4.3	0.015	1.8
Screen 100. .	11.0	0.36	3.2	0.010	0.8	13.2	0.20	5.4	0.015	2.2
Screen 150	16.0	0.42	4.1	0.013	0.8	15.0	0.20	5.4	0.02	2.1
Through 150.	21.0	0.48	9.0	0.015	1.3	15.2	0.26	13.2	0.03	3.0

TABLE X.—*Sizing-Tests of Sand, Using Different Battery-Screens.*

Battery-Screen.	Quantity by Weight. On 40.	On 60.	On 80	On 100	On 150.	Through 150
	Per Cent.	Per Cent.	Per Cent	Per Cent	Per Cent	Per Cent.
0.6-mm. slot .	10.0	20.0	17.0	13.0	15.0	25.0 (21.0 sand, 4.0 slime).
1.0-mm. needle.	11.7	19.5	19.2	11.2	19.7	18.7 (16.0 sand, 2.7 slime)
0.6-mm needle	9.0	21.0	17.0	11.0	16.0	26.0 (21.0 sand, 5.0 slime).
0.6-mm tin	5.0	19.3	12.6	13.3	19.0	30.3 (24.0 sand, 6.0 slime) [a]
0.6-mm. tin ..	8.5	24.5	22.0	7.0	16.0	22.0 (14.0 sand, 8.0 slime).
0.6-mm tin ...	5.0	18.5	17.6	16.6	32.3	10.0 (8.5 sand, 1.5 slime).

[a] Loss of 1.5 per cent. in screening.

TABLE XI.—*Results of a Typical Treatment of Sand.*

Charge, 145 tons of sand. Collected, 9 N 6.[b]

Residue: gold, 0.26 oz.; silver, 7.1 oz. per ton. Drained from 9 N 6 to 4 N 10.

Undissolved: gold, 0.18; silver, 5.7 oz. per ton.

Dissolved: gold, 30.7; silver, 19.7 per cent. Transferred to treatment-tank, 12 D 10.

Solution added three times daily. The quantity given is the sum of the additions.

Day.	Solution: Quantity	Solution: Strength: KCN.	Solution: Strength: CaO.	Leachings: KCN.	Leachings: CaO.	Leachings: Gold.	Leachings: Silver.	Undissolved: Gold.	Undissolved: Silver.	Residue: Gold.	Residue: Silver.
	Tons	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Oz. per Ton.	Oz per Ton.	Oz per Ton.	Oz. per Ton	Oz per Ton.	Oz. per Ton.
1	44	0.23	0.01	0.12	0.08	0.256	5.35	0.12	4.0	0.19	5.5
2	43	0.20	0.03	0.15	0.08	0.180	4.50	0.06	3.3	0.13	4.3
3	46	0.25	0.05	0.15	0.07	0.120	2.77	0.04	2.2	0.09	3.0
4	47	0.20	0.08	0.17	0.07	0.07	1.60	0.04	1.8	0.08	2.4
5	40	0.15	0.03	0.20	0.08	0.03	1.00	0.03	1.7	0.06	2.3
6	Charge aerated.										
7	Charge aerated.										
8	Charge aerated.										
9	41	0.13	0.09	0.13	0.03	0.05	1.70	0.02	1.5	0.04	1.7
10	44	0.14	0.09	0.12	0.07	0.028	0.80	0.02	1.4	0.03	1.5
11	44	0.18	0.09	0.12	0.07	0.020	0.50	0.01	1.4	0.03	1.4
12	44	0.19	0.09	0.11	0.08	0.014	0.44	0.015	1.2	0.03	1.2
13	Charge aerated.										
14	Charge aerated.										
15	35	water	wash	0.11	0.08	0.019	0.48	0.02	1.0	0.02	1.0
16	12	water	wash	0.07	0.07	0.01	0.50	0.015	0.9	0.02	1.1
17	35	water	wash	0.04	0.06	0.005	0.20	0.015	1.0	0.01	0.9
18	23	water	wash	0.02	0.07	0.002	0.06	0.015	1.0	0.02	1.0
								Recovery......		Per Ct. 92.3	Per Ct. 85.9

[b] N and D are used as abbreviations instead of a.m. and p.m. N is that time between 6 p.m. and 6 a.m. D between 6 a.m. and 6 p.m. The date is changed at midnight.

The slime was separated from the product finer than 150-mesh by washing and settling, that portion being taken as sand that would settle and leave the solution clear in 30 seconds.

A summary of the cost per ton of sand treated for a period of seven months, based on a total quantity of 23,925 tons, is given in Table XII.

TABLE XII.—*Cost Per Ton of Sand Treated.*

Labor (native),	$0.1066
Employees (white),	0.0324
Cyanide,	0.2367
Lime,	0.1040
Filter-cloths,	0.0074
Shovels,	0.0028
Pipe and sundries,	0.0025
Miscellaneous,	0.0130
Power,	0.0934
Total,	$0 5988

2. *Slime-Plant.*

The slime-plant consists of two collecting-tanks, 35 ft. in diameter by 9 ft. 6 in. deep, and ten treatment-tanks, 24 ft. in diameter by 9 ft. 6 in. deep, with filter-presses, necessary pumps, etc. All the tanks are of redwood with 3-in. staves.

The treatment-tanks are arranged in two lines of five tanks each (Fig. 7). Each tank is equipped with a mechanical agitator consisting of two cross-arms, driven by crown-wheel and spur-gear from a line-shaft over the top of the tanks. The friction-clutch (Fig. 8) used in driving each gear has proved efficacious in saving gears, is simple, and does not get out of order. There has been but one gear replaced in two years. The arms revolve six times per minute.

There is a 4-in. centrifugal pump connected to each tank. These pumps are driven by friction-clutch pulleys from a line-shaft between the tanks, running at 800 rev. per min. They handle the contents of a tank in 1 hr. 15 min. The pumps have solution-lubrication, and the runners and shaft last for two years. The runners, of the inclosed type, have given no difficulty in pumping thick or sandy slime, and have a much greater efficiency than open runners of the star type. Each pump requires 1.5 and each agitator 1 horse-power.

The agitator-arms, 6 by 8 in. in size, have 1-in. cast-iron rods

driven through them and projecting 6 in. These rods protect the bottom of the arms from wear by the sand settling in the bottom of the tank. The vertical shaft, which drives the cross-arms, is set in a step-box bolted to the bottom of the tank, and the casting to which the cross-arms are bolted is faced to fit

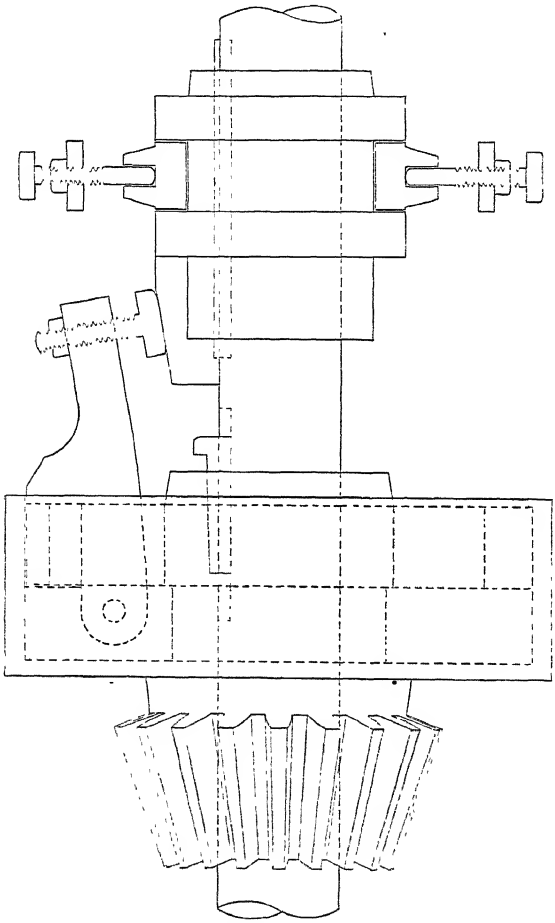

FIG. 8.—SPECIAL CLUTCH OF HOME-MADE CONSTRUCTION.

the step-box. In this way no sand is able to enter the box, and all the boxes examined to date are in as good condition as when first installed.

The power for the slime-plant is supplied by an 80-h-p. slide-valve engine, which runs the agitators, the treatment-pumps, and the centrifugal pumps for the batteries and storage-tanks.

The pulp overflowing from the sand-collecting tanks is delivered by a 6-in. pipe, set on a 1-per cent. grade, to a box at the side of one of the 35-ft. slime-collecting tanks. At the opposite side of the tank is a slotted decanting-pipe, through which the overflow of solution passes to the second tank. From the second tank a similar pipe carries the solution to the battery-sump, whence it is returned to the battery-supply tank by the centrifugal pumps or a 6-in. steam-pump.

The flow of solution from the batteries is approximately 60 tons per hour. With the two collection-tanks in series a clean overflow of solution is possible with the most slimy ore.

Slime-charges are drawn from the collecting-tanks, at intervals depending on the time of treatment and the tanks available, by a 4-in. centrifugal pump running 1,200 rev. per min. This pump transfers a charge of 35 tons in 30 min. During the transference of a charge the agitators in the collection-tanks are revolved to prevent the slime from packing in the tank; these agitators are similar to those in the treatment-tanks, but turn 2 rev. per minute.

The charge, after transference from the collecting-tanks, is agitated with pump and stirrer, settled, and decanted. This operation is repeated, using weak precipitated solution. The number of treatments given varies with the ore. Practically all the gold is dissolved in from 4 to 6 hr. after the first agitation begins, while the silver dissolves slowly and quite uniformly whether the pulp is being agitated or not. For this reason short agitations are given and as much time as possible is devoted to settlement and decantation, the object of the agitations being chiefly to mix thoroughly the slime and solution.

The addition of oxygen from the air, necessary for the dissolution of metals by cyanide, is much better made to the clean solution before it is mixed with the slime, for it is much more soluble in clean solution than in a thick pulp, and therefore frequent additions of fresh solution are preferable to continued agitation of a thick pulp. This method has been found to give the best results, even after the necessity for thorough washing has been obviated by the installation of the filters.

Clean unoxidized ore, settled 12 hr., contains 55 per cent. of moisture. Mixed ore containing from 30 to 40 per cent. of oxi-

dized ore, settled the same time, contains from 60 to 70 per cent. of moisture. The presence of from 5 to 10 per cent. of fine sand in the slime greatly increases the rate of settlement, and at the same time decreases the amount of moisture in the pulp. Pulp during agitation has a specific gravity of from 1.18 to 1.21; after settlement and decantation it runs from 1.24 to 1.28.

There are ten tanks available for the treatment of slime; one is used as a supply-tank for the filters, and of the nine remaining, two are at present used for the treatment of slime accumulated during the last years of the pan-amalgamation plant. With the seven tanks remaining, 70 hr. are available for treatment, charging a tank every 10 hr. Four agitations are given, the first of 8 hr. and the three following of 4 hr. each, which allows time for four 12-hr. decantations and 2 hr. for the charging of the tank and the transference of the charge to the filter-supply tank. The transference is done by the same pump used in charging the treatment-tanks.

The results of the treatment of a typical charge of mixed ore in the slime-plant are shown in Table XIII. The data cover three periods, in which the ore and treatment have varied. In the first period, that of 1907, Table XIV., clean unoxidized ore was treated and weak solutions were used. In the second period,

TABLE XIII.—*Results of a Typical Treatment of Slime.*

Dry slime charged, 35.5 tons. Specific gravity of the pulp charged, 1.2.
Solution in charge, 92 tons.
Agitation by means of pump and stirrer.
Solution added at beginning of each treatment is precipitated solution.

Agitation.							Decantation					
			Assay Pulp, Beginning of Agitation.						Assay Decanted Solution.		Assay Pulp, End of Agitation	
Time.	Solution Strength.		Residue.		Undissolved.		Time.	Quantity.			Undissolved.	
	KCN.	CaO.	Gold.	Silver.	Gold.	Silver.			Gold.	Silver.	Gold.	Silver.
Hr.	Per Cent.	Per Cent.	Oz. per Ton.	Oz. per Ton.	Oz per Ton.	Oz. per Ton.	Hr.	Tons.	Oz. per Ton.	Oz per Ton.	Oz. per Ton.	Oz. per Ton.
8	0.09	0.07	0.25	8.8	0.14	5.7	12	27	0.085	2.5	0.04	2.3
4	0.10	0.05	0.19	7.0	0.02	1.7	12	28	0.060	2.4	0.04	1.5
4	0.11	0.04	0.13	5.2	0.01	1.4	12	30	0.045	1.4	0.01	1.3
4	0.10	0.04	0.08	3.8	0.01	1.5	14	32	0.026	1.0	0.01	1.1
Discharge to filters			0.04	2.8	0.01	1.1	..	..	0.027	1.0		

TABLE XIV.—*Results of Operations at the Slime-Plant.*

1907, Month	Dry Slime.	Solution Ratio	Strength of Solution		Charging		Discharge		Extraction	
			KCN	CaO	Gold.	Silver.	Gold.	Silver.	Gold	Silver.
	Tons		Per Cent.	Per Cent	Oz per Ton	Oz. per Ton.	Oz per Ton	Oz per Ton	Per Cent	Per Cent
March .	1,677	6.0	0.130	0.018	0.434	10.7	0.01	1.06	97.7	90.07
April.	1,640	5.3	0.135	0.027	0.291	9.1	0.01	1.08	96.3	86.7
May .	1,766	5.2	0.124	0.030	0.337	7.3	0.01	1.12	98.7	84.6
June	1,686	6.8	0.103	0.014	0.291	5.7	0.01	0.08	96.5	85.8
July	2,065	5.4	0.098	0.006	0.308	5.1	0.01	0.96	96.7	81.3
August.	2,197	5.0	0.106	0.006	0.144	4.8	0.012	1.53	91.4	72.2
Total	11,031	.	.	...	0.294	6.83	0.01	1.11	96.6	83.6
		With silver at $0.50			$9.49		$0.76		91.9 Per Cent.	
1908										
Feb.	2,115	3.91	0.109	0.013	0.240	6.71	0.01	2.05	91.9	69.3
March...	2,206	4.89	0.113	0.020	0.402	7.15	0.01	2.07	97.6	71.0
April .	2,459	4.5	0.124	0.019	0.318	6.56	0.02	1.65	93.6	74.7
May	2,307	5.5	0.099	0.007	0.272	6.90	0.01	2.02	96.3	70.6
June .	2,142	5.31	0.097	0.008	0.338	7.32	0.02	1.89	94.0	74.1
July	2,393	5.0	0.086	0.009	0.339	8.05	0.025	2.26	92.5	72.0
Total .	13,652	....	.	. .	0.318	7.12	0.0.6	1.99	94.9	72.0
		With silver at $0.50			$10.13		$1.32		86.9 Per Cent.	
1909.										
March...	2,648	4.0	0.191	0.029	0.246	7.6	0.010	1.15	95.9	84.8
April...	2,878	3.8	0.156	0.036	0.275	7.2	0.010	1.23	96.3	84.7
May	2,853	4.0	0.144	0.045	0.222	6.9	0.011	1.26	95.0	81.7
June ...	2,747	4.0	0.108	0.033	0.262	7.6	0.012	1.30	95.4	82.8
Total ...	11,126	....	..	.....	0.251	7.4	0.011	1.24	95.6	83.2
		With silver at $0.50			$8.88		$0.84		90.5 Per Cent.	

that of 1908, a large amount of unoxidized ore was treated, the strength of the solutions remaining the same. The last period is for 1909, oxidized ore being treated, but with a solution stronger in cyanide. In this latter period the work of the filters is shown.

A comparison of the cost of slime-treatment before and after the installation of filters is given in Table XV.

3. *Filters.*

The filter-plant has two Oliver continuous filters, each of 60 tons capacity. These filters consist of drums, 11 ft. 6 in. in diameter by 8 ft. wide, revolving on horizontal axes in boxes or tanks. The outer surface of the drums is divided into sec-

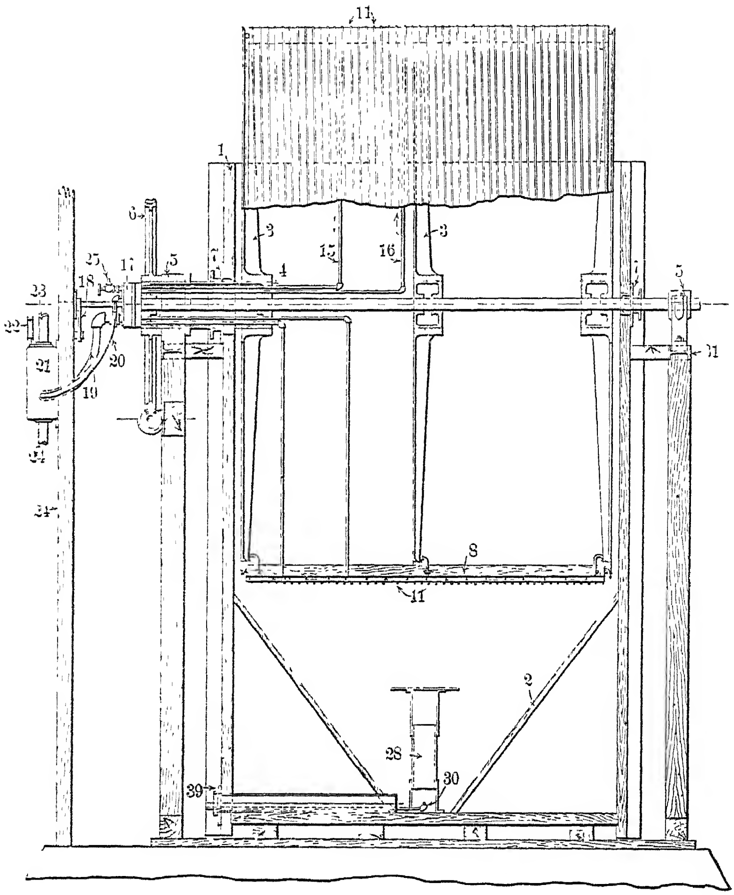

FIG. 9.—OLIVER CONTINUOUS FILTER. TRANSVERSE SECTION AND ELEVATION.

1. Redwood tank.
2. False bottom.
3. Cast-iron spiders supporting filter-drum.
4. Hollow trunnion carrying spiders, piping, etc.
5. Main bearings.
6. Worm-drive gear.
7. Stuffing-boxes.
8. Redwood staves for filter-drum.
9. Filter medium.
10. Division-strips dividing filter into sections.
11. Steel-wire protection for filter medium.
12. Steel scraper.
13. Adjusting-screw and lever for scraper.
14. Tailings-apron.
15. Vacuum-pipes from filter-sections to automatic valve.
16. Pressure-pipes from filter-sections to automatic valve.
17. Automatic valve.
18. Adjusting-lever for valve.

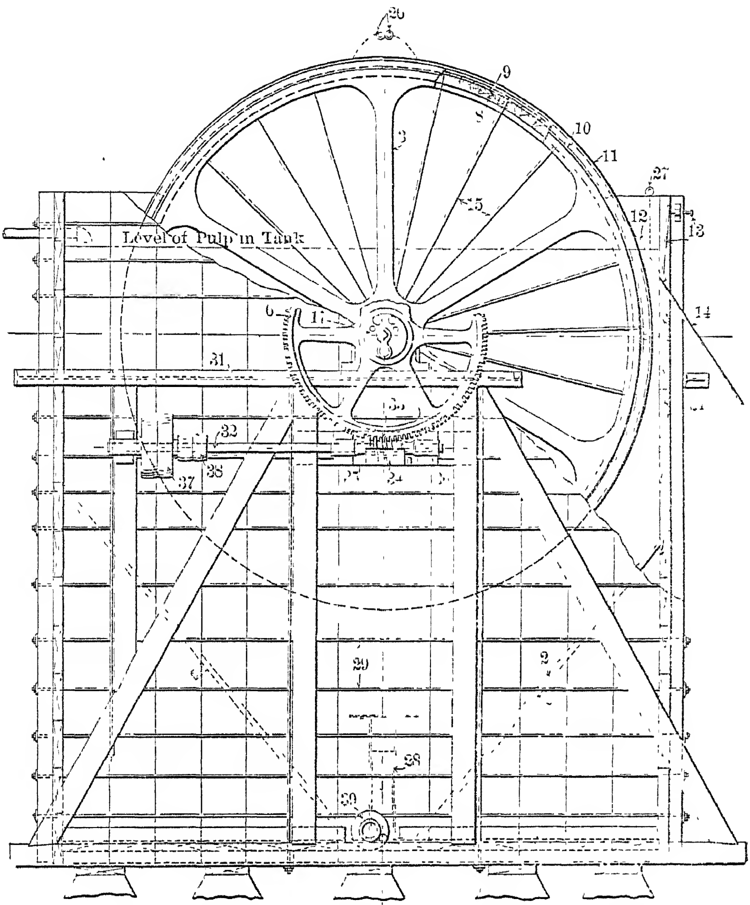

FIG. 10.—OLIVER FILTER. LONGITUDINAL SECTION AND ELEVATION.

19. Flexible vacuum-hose for working-solution.
20. Flexible vacuum-hose for wash-solution.
21. Vacuum-chamber containing float check-valve.
22. Vacuum-gauge.
23. Pipe to dry-vacuum pump.
24. Pipe to solution-pump.
25. Pressure-pipe for blowing filter-sections.
26. Wash spray-pipe.
27. Discharge spray-pipe.
28. Agitator-pipe.
29. Tie-rods.
30. Agitating air-nozzle.
31. I-beam frame.
32. Worm-shaft.
33. Worm.
34. Oil-well for worm.
35. Thrust-collars for worm.
36. Post-bearings for worm-shaft.
37. Driving-pulleys.
38. High-speed pulleys for wiring.
39. Flange for drain.

TABLE XV.—*Cost of Treatment of Slime.*

Item	Before Installation of Filters. Cost Per Ton of Slime Treated.	After Installation of Filters, Cost Per Ton of Slime	
		Slime-Plant	Filters
Native labor........	$0.0803	$0.0890	$0.0372
Supervision...	0.0513	0.0497	0.0215
Cyanide...................... .	0.6007	0.3434	....
Lime.......	0.1710	0.1035	....
Pumps, piping......	0.0036	0.0013	0.0012
Belts, lacing	0.0085	0.0084	0.0067
Lubricant......................	0.0088	0.0022	0.0033
Miscellaneous	0.0364	0.0099	0.0066
Total..................... ..	$0.9606	$0.6074	$0.0765
Power.....	0 2534	0.1419	0.0835
		0.7493	0.1600
Total..........	$1.2140	$0.9093	
Tons treated..	17,540	11,126	

tions and covered with the filtering-medium. The drums are partly submerged in the pulp to be filtered (kept from settling by air-agitation) and, as they revolve, a vacuum applied to the sections builds up a cake of slime on the surface. As the sections leave the pulp they are dried by the vacuum and then a water-wash is applied and drawn through the slime. Just before the section enters the pulp again, air, under pressure, is admitted to the chamber and the cake is discharged. The cycle is then repeated, the whole work being continuous and automatic.

The construction of the filters is shown in Figs. 9 and 10. The drum is built of 3.5-in. staves firmly bolted to the spiders. The perimeter of the drum is divided into 24 sections by 1-in. strips. Small strips, 0.5 by 0.5 in., nailed 1 in. apart, form channels for the passage of solution in each section. Two 0.5-in. pipes are connected to each section, which pass through the hollow shaft supporting the drum to a plate at the end.

The plate has two circles of 24 holes each, to which the pipes are connected. Facing this plate is a second plate, having a groove or channel opposite the outer circle of holes. An adjustable bridge in this groove covers one hole, so that, the channel being connected to the vacuum-pumps, a vacuum is

applied to 23 of the sections. Air under pressure, passing through the pipe connected to the inner circle of holes in the plate, is admitted to the section that is cut off from the vacuum by the bridge. As the drum revolves the vacuum is cut off from each section in turn and air is admitted.

A screen is placed on the strips forming channels in the sections. The strips dividing the drum into sections are planed level with the surface of the screens. A layer of burlap covers the screens and on this the canvas is placed. The canvas is drawn tight and calked into a groove around the edge of the drum. To bind the canvas firmly to the division-strips, the drum is wound with spring-steel wire. The wiring makes each section tight and prevents the canvas from bulging when the cake is discharged.

The drums are submerged for three-fifths of a revolution. Every section as it enters the pulp is connected to the vacuum-pumps. As the drum revolves the section picks up a cake of slime and, upon leaving the pulp, this cake is dried by the vacuum. A water-wash is then applied in the form of a drip, in much the same manner as clean water is added in cleaning the concentrate on a vanner. This water is drawn through the cake until the vacuum is cut off and the cake discharged. The discharge takes place just before the section enters the pulp for a repetition of the cycle. The drums revolve once every 4 min.

The air and solution from the sections go to a chamber that is connected to the wet- and dry-vacuum pumps. Here the solution and air are separated, the wet-vacuum pump being connected to the base of the chamber, the dry-vacuum pumps to the top.

Two one-cylinder dry-vacuum pumps and one double-cylinder solution-pump are used. (These wet- and dry-vacuum pumps have been displaced by a wet-vacuum pump which maintains a 26-in. vacuum and takes less power.) A small compressor furnishes air at from 5 to 10 lb. pressure to discharge the slime and agitate the pulp in the filter-boxes. To operate the pump, compressor, and drums, 13 h-p. are required. With this equipment the capacity of the filters is 125 tons of dry slime per day.

The vacuum, maintained at 25 in., yields a cake of slime varying from $\frac{3}{16}$ to $\frac{3}{8}$ in. thick, depending on the character of

the slime filtered. Between the time a section leaves the pulp and the application of the water-wash, the slime is dried to 35 per cent. of moisture. The amount of water drawn through the cake is practically equal to the moisture in the slime. As the pulp ordinarily filtered has a specific gravity of 1.24, the amount of solution passed through the filters is large, the ratio of solution to slime in the pulp being 2.3 to 1.

In treating oxidized ore, 33 per cent. of moisture is discharged in the residue, which is chiefly clay having the character of a colloid. A certain quantity of tailings from the former pan-amalgamation plant has been re-treated in the slime-plant. These tailings contained a large amount of fine sand from the pans, and the filters reduced the moisture in this material to 25 per cent.

The water-wash almost completely displaces the gold, silver, and cyanide in the cake. The quantity of cyanide discharged is from 0.1 to 0.3 lb. per ton of dry slime. The undissolved content of the slime filtered is practically equal to the content of the discharged residue, the difference between the two being 0.1 oz. of silver.

The lime in the solutions gradually deposits in the cloth and cuts down the capacity of the filters. To rectify this, the cloth is washed once every two weeks with dilute hydrochloric acid. The washing takes an hour, and 7 lb. of acid is used in treating one drum.

There is 290 sq. ft. of filter-surface on each drum. Filtering 60 tons per day, per drum, each square foot handles 400 lb. of dry slime. The canvas lasts from three to five months. To re-cover and re-wire a drum takes from 12 to 18 hours.

The cost of re-covering a drum is:

13 yd. No. 12 duck, 104 in. wide.		$14.48	
25 yd. 8-oz. burlap,		2.65	
77 lb. No. 17 spring-steel wire,		6.70	$23.83
Mechanic and 3 *peones*, 16 hr.,			10.00
Total,			$33.83
Tons filtered by cloth,	4,000		
Cost per ton filtered,			$0.0084

One *peon* attends the filters and machinery, an expense which would not be necessary in the United States, where labor is efficient, since the greater part of the time no attention is needed beyond the oiling of the machinery.

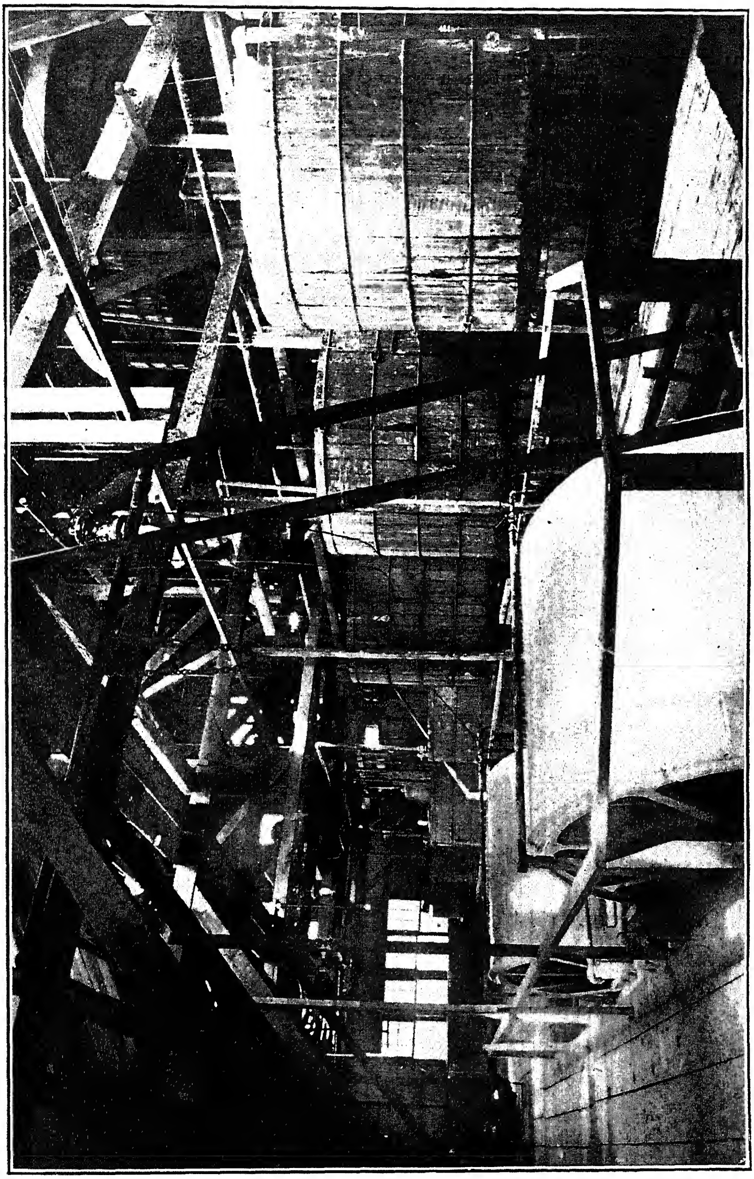

FIG. 11.—THE 50-TON OLIVER FILTER OF THE NORTH STAR MINES CO., GRASS VALLEY, CAL.

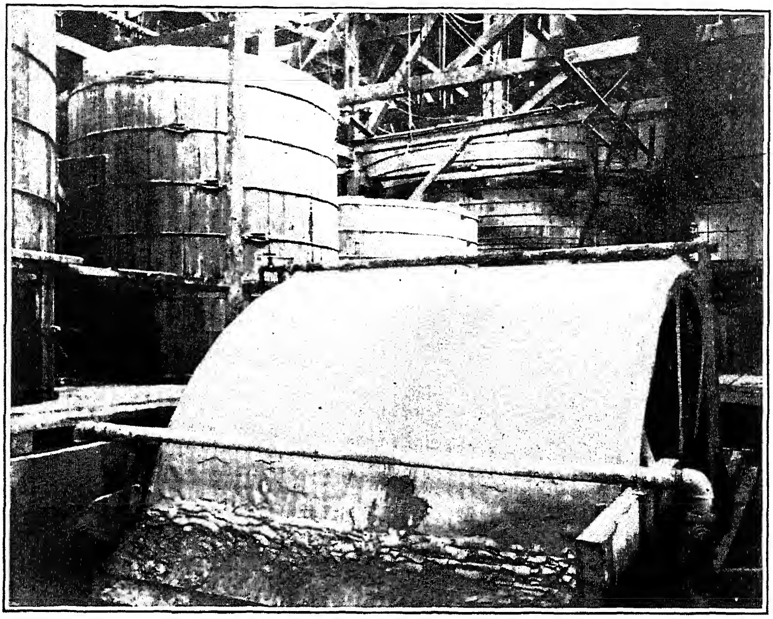

FIG. 12.—THE 50-TON OLIVER FILTER OF THE NORTH STAR MINES CO., GRASS VALLEY, CAL., SHOWING THE REMOVAL OF CAKED SLIME BY SCRAPER.

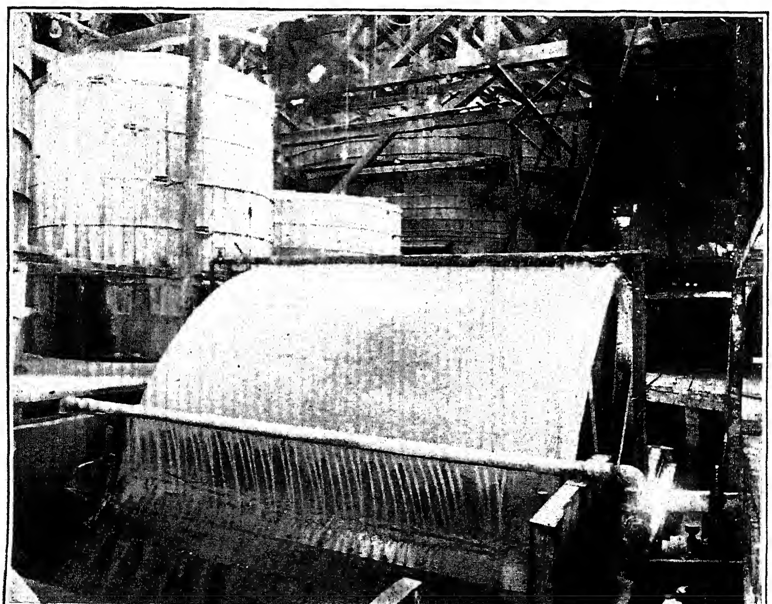

FIG. 13.—THE 50-TON OLIVER FILTER OF THE NORTH STAR MINES CO., GRASS VALLEY, CAL., SHOWING THE REMOVAL OF CAKED SLIME BY WATER-SPRAY.

The cost of the operation of the filters given in connection with the slime-plant includes a proportion of the charges for supervision of mill, shiftmen, time-keeper, watchmen, and shop-charges for mill. The direct cost of operation per ton of dry slime filtered, not including pro rata supervision, etc., is:

Labor,	$0.0161	
Covering drum, material,	0.0060	
Covering drum, labor,	0.0022	
Repairs and supplies,	0.0112	
Lubricant,	0.0033	$0.0388
Power,		0.0835
Total,		$0.1223[a]

[a] Filters running under capacity.

In Table XVI. the results of operations at the filtering-plant for a period of four months are given. During this time the amount of slime filtered was less than the capacity of the filters.

TABLE XVI.—*Results of Operations at the Filtering-Plant.*

Quantity of Dry Slime.	Charging.		Solution Recovered.				Discharge Residue.	
	Undissolved Silver.	Residue. Silver.	KCN.	CaO.	Gold.	Silver.	Gold.	Silver
Tons.	Oz. Per Ton.	Oz. Per Ton	Per Cent.	Per Cent.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton.	Oz. Per Ton.
2,648	1.10	2.13	0.152	0.02	0.020	0.83	0.010	1.15
2,878	1.25	2.49	0.128	0.02	0.023	1.01	0.010	1.23
2,853	1.09	2.78	0.117	0.03	0.025	1.03	0.012	1.24
2,747	1.17	2.80	0.086	0.02	0.026	1.06	0.012	1.30
11,126	1.15	2.55	0.120	0.02	0.023	0.98	0.011	1.24
							$\underbrace{\qquad\qquad}_{\$0.85}$	

Figs. 11, 12, and 13 are photographic views of an Oliver 50-ton continuous filter operated by the North Star Mine Co., Grass Valley, Cal., which illustrate the manner of removing the slime from the drum.

4. *Precipitation.*

The solutions from the sand- and slime-plants are collected in sumps situated at the foot of the plant. Centrifugal pumps deliver the solution for precipitation to two tanks placed above the precipitation-room, which is 40 by 80 ft. in area. A plan of

this room, showing the position of the zinc-boxes, furnaces, etc., is given in Fig. 14. Along one side of the room are zinc-precipitation boxes of two types, arranged as shown on Fig. 15; 8 are of sheet steel, having 10 compartments, 12 by 22 by 27 in.; 6-mesh screens are placed 5 in. above the bottom. The bottoms are hopper-shaped and discharge through 2-in. iron cocks to the clean-up launder, which delivers the solution and precipitate to the sump situated in the middle of the room on the side containing the boxes.

Two pairs of wooden boxes are placed on the other side of the sump, each having five compartments, 24 by 24 by 22 in., and one compartment, 12 by 24 by 22 in., used for settling. The

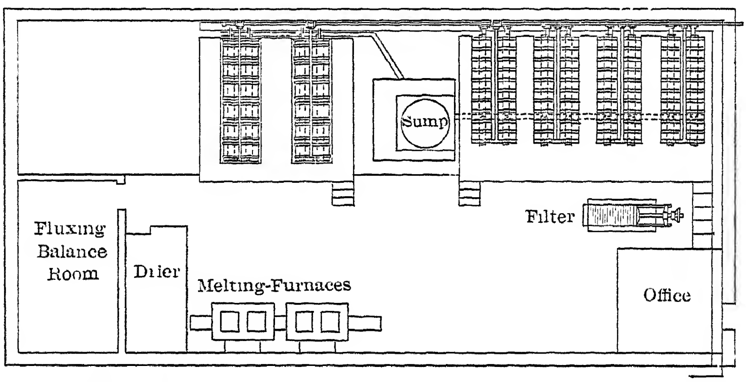

Fig. 14.—Plan of Precipitation-Room, Showing the Position of Zinc-Boxes, Furnaces, Etc.

boxes have one side in common. The bottom slopes to the side, the precipitate being delivered to the sump by a launder along the side of the box. These boxes have 4-mesh screens placed 5 in. above the bottom. Both sets of boxes are set 3 ft. above the concrete floor, which is drained to a sump. The precipitated solutions flow to two tanks at the side of the building.

The precipitate is delivered on to a screen over the clean-up tank, which is of redwood, 6 ft. in diameter by 4 ft. deep, and is fitted with a canvas filter. This tank drains into the sump, above which it is placed. The solution obtained from the clean-up is pumped by a 2-in. steam-pump to a 24-in. Shriver filter-press, holding 24 frames, each 1 in., the effluent solution from which goes to the head of the low-grade extractors. The

precipitate is settled in the tank, drained over night, and put into the drier.

The drier consists of a steel box with cast-iron plates for a base, resting on a flue shaped like the letter U. The flames from a small fire-box at one end thus pass twice the length of the drier before reaching the stack. A small fire serves to dry from four to eight trays of precipitate, weighing from 440 to 1,000 lb., in five hours.

On the side of the room facing the precipitation-boxes are two pairs of melting-furnaces, each 18 in. in diameter and 26 in. deep. There is a stack for each pair of furnaces, natural draft alone being used.

Two grades of solution are precipitated. The high-grade solution, consisting of the first drainings from sand-plant, runs from 3.5 to 4.5 oz. of silver and from 0.15 to 0.25 oz. of gold per ton. Six of the steel boxes are used to receive this solution. Nine of the 10 compartments are filled with zinc; the last compartment being used as a settler. Each set of boxes contains 30 cu. ft. of zinc-shavings. The rate of flow of solution has varied from 1.1 to 2.7 tons of solution per 24 hr. per cubic foot of zinc; the precipitation is good at a rate up to 3 tons per cubic foot of zinc; above this rate it falls off.

The low-grade solution comprises the intermediate washes from the sand and the first decantations from the slime. These solutions carry from 1 to 2 oz. of silver and from 0.03 to 0.06 oz. of gold per ton. The solution coming from the slime-plant was until recently sent to the extractors without filtering. This gave no trouble and the precipitate was of good grade; but in order to avoid a slow accumulation of slime in the sump the solution is now filtered.

The low-grade solutions are precipitated in two of the steel and two of the wooden boxes. The wooden boxes are more convenient and easier to clean. The size of the compartments is more suitable for a silver solution than the smaller ones of the steel boxes. The rate of flow for the wooden boxes has varied from 4 to 8 tons of solution per 24 hr. per cubic foot of zinc. The precipitation takes place almost entirely in the first two compartments, the solution having ample cyanide for good precipitation.

An interesting feature of the work is the regeneration of

cyanide. As will be seen from Tables XVII. and XVIII., the cyanide-strength of the solution leaving the boxes is greater than that of the entering solution. Drip-samples, taken at the head and foot of the boxes, have shown a regeneration of cyanide for a period of two years, due to the lime present, which reacts with the zinc in solution, forming zinc hydrate and liberating the cyanide.

TABLE XVII.—*Results of Precipitating High-Grade Solutions.*

(Monthly Averages.)

Date.	Incoming Solution				Outgoing Solution.				
	KCN.	CaO.	Gold.	Silver.	KCN.	CaO.	Gold.	Silver.	Silver Precipitate.
	Per Cent.	Per Cent.	Oz Per Ton.	Oz. Per Ton.	Per Cent.	Per Cent	Oz. Per Ton.	Oz Per Ton.	Per Cent
Sept., '08.........	0.137	0.015	0.150	3.19	0 150	0.020	0.002	0.020	99.3
Oct., '08...........	0.136	0.038	0.182	3.57	0.154	0.059	0.002	0.017	99.5
Nov., '08..........	0 143	0.074	0.179	3.40	0.161	0.083	0.003	0.020	99.4
Dec., '08..	0.124	0.069	0.210	3.76	0.142	0.071	0.002	0.033	99.1
Feb., '09.........	0.184	0.027	0.155	2.82	0.192	0.031	0.001	0.040	97.9
Mar., '09.........	0.215	0.025	0.163	3.69	0.224	0.025	0.002	0.047	98.7
Apr., '09..........	0.210	0.047	0.161	4.46	0.222	0.052	0.002	0.080	98.2
May, '09..........	0.170	0 052	0.153	4.13	0.181	0.060	0.002	0.032	99.2

TABLE XVIII.—*Results of Precipitating Low-Grade Solutions.*

(Monthly Averages.)

Date	Incoming Solution.				Outgoing Solution				
	KCN.	CaO.	Gold.	Silver.	KCN.	CaO.	Gold	Silver.	Silver Precipitate.
	Per Cent	Per Cent.	Oz. Per Ton	Oz. Per Ton.	Per Cent.	Per Cent.	Oz. Per Ton	Oz. Per Ton.	Per Cent.
Sept., '08	0.102	0.021	0.027	0.81	0.103	0.020	0.001	0.05	93.8
Oct., '08...........	0.103	0.026	0.037	1.01	0.103	0.027	0.001	0.05	95.0
Nov., '08	0.125	0.035	0.040	1.01	0.128	0.031	0.001	0.06	94.0
Dec., '08...........	0.105	0.028	0.035	0.90	0.110	0.029	0.001	0.04	95.5
Feb., '09..........	0.155	0.023	0.044	1.00	0.160	0.024	0 001	0.09	91.0
Mar., '09..........	0.167	0.032	0.040	1.14	0.191	0.029	0.001	0.13	88.5
Apr., '09..........	0.156	0.035	0.034	1.44	0.156	0.036	0.001	0.17	88.2
May, '09	0.143	0.044	0.033	1.40	0 147	0.045	0.001	0.09	93.5

The slight variation of the plant-solution after two years' use is shown in Table XIX. The first analysis was made after the plant had been running three weeks and the solutions were fresh; the others follow at intervals of six months. For all practical purposes the solutions have remained the same. The in-

crease in oxidized ore carrying organic matter accounts for the increase in reducing-power.

TABLE XIX.—*Composition of the Plant-Solution.*

Date.	March, 1907		Sept, 1907		Sept., 1908		March, 1909	
	A.	B.	A	B	A	B.	A.	B
Gold, oz. per ton........	0.120	0.001	0.158	0.002	0.150	0.002	0.163	0.002
Silver, oz. per ton.......	3.06	0.05	3.05	0.02	3 19	0 02	3.69	0.04
KCN, per cent...........	0 180	0.220	0.120	0.130	0.137	0.150	0.215	0.224
Protect. CaO, per cent	0.024	0.040	0.035	0.042	0.015	0.020	0.025	0.025
Total CaO, per cent....	0.132	0.164	0.105	0.105	0.103	0.124	0.180	0.210
Reducing-power... ...	0.730	0.860	0.540	0.600	0.340	0.313	0.619	0.561
Ferrocyanide, per cent.	0.018	0.022	0.060	0.072	Nil.	Nil.	Nil.	Nil.
Copper, per cent.........	0.038	0.018	0.038	0.017	0.014	0.002	0.019	0.008
Zinc, per cent....... ...	0.065	0.084	0.036	0.045	0.027	0.045	0.038	0.068

A. Strong plant-solution after passing through the sands.
B. Same solution after precipitation.

During 1907 caustic soda was used as the neutralizing agent, and coincident with its use there developed a considerable amount of ferrocyanide in the solution. Upon changing to the use of lime alone the amount of ferrocyanide dropped and is now *nil*. Furthermore, lime prevents the fouling of the solution by iron, precipitating it as ferric hydrate. It also has the advantage of being a good settling-agent for the slime.

The boxes are cleaned up each week. Half the strong and half the weak boxes are cleaned one day, the remainder three days later. The boxes are all dressed with zinc on clean-up days. Two natives attend to this work. The flow of solution being stopped, the coarse zinc remaining in the head-compartment is rinsed in the compartment and transferred to a tub of water. When the coarse zinc has been removed, the solution and precipitate are discharged into the launder. A layer of new zinc is placed on the screen and the washed zinc returned. The compartments are cleaned in rotation, the zinc removed going first to fill the head-compartment, and then the others in order. When all the zinc has been returned, there remain two or three compartments empty, which are filled with fresh zinc and the flow of solution started. The overflow is clear on starting the boxes.

The fine zinc and precipitate is delivered by the launder to a No. 40 brass screen over the clean-up tank. The fine precipitate passing the screen settles on the filter-cloth and the

rate of filtration decreases. To avoid delay from this cause, the excess solution arising from the washing of the zinc is pumped from the top of the tank to a filter-press.

The fine zinc remaining on the screen is well washed with water and returned to the head-compartments of the zinc-boxes. In two years there has been no accumulation of short zinc.

The precipitate in the tank is drained over night and the following morning is placed in the drier. About 95 per cent. of the clean-up is obtained from the tank, the remainder passing to the filter-press. The press is cleaned bi-monthly, the product amounting to about 250 kg. An hour is sufficient to clean the tank and place the product in the drier. To clean the press properly takes from 4 to 6 hr. A set of cloths for the press costs $26, while the filter in the tank costs $3. The durability is approximately the same, from eight to ten months. The amount of precipitate obtained each week is approximately 1,000 pounds.

When all solutions were filtered before precipitation the percentage of fine metal in the precipitate was 69.2 per cent. for a period of five months. The average of the succeeding five months, during which period 20,000 tons of solution were sent to the extractors without filtering, was 66.6 per cent.

5. *Melting.*

Four furnaces, 18 in. in diameter by 26 in. deep, are used for melting the precipitate, using No. 80 Dixon and No. 100 Battersea crucibles. The fuel is a mixture of 3 parts of charcoal to 1 of coke. Natural draft is used, giving ample heat. The crucibles are handled with a lever hung from an overhead track serving the four furnaces. Two natives, working from 12 to 16 hr., flux and melt the week's clean-up into bars.

The dried precipitate is broken up and bedded on a zinc-lined table and the flux added. The flux consists of: ground borax-glass, 20; assay-slag, 6, and sand, from 2 to 4 per cent. The fluxed precipitate is put in manila-paper bags for charging into the pots.

The fires are started the first thing in the morning, and after the crucibles are hot they are filled with the precipitate. As the charge subsides more is added till the charge is fluxed.

This usually takes from an hour to an hour and a half. A melt requires an additional hour. The pours are made into conical molds, the slag obtained being practically as glassy as the average assay-slag and free from shot.

The melts are arranged to give from 15 to 20 kg. of bullion, two melts going to make a bar. The bullion from the cones has a thin skin of matte, soft and malleable. The cones of bullion are remelted with borax-glass and iron, the amount of iron used depending on the quantity of matte on the bullion. The iron serves to make a matte which is easily removed from the bars, and at the same time reduces the value of the matte by replacing some of the silver. The melted bullion is skimmed till clean, thoroughly stirred, two dip-samples taken, and then poured into molds. The bars are cleaned with dilute sulphuric acid. The average fineness of gold and silver during 1908 was 932.

The dip-samples are bored and the borings used for the assays. Much more reliable and concordant results are obtained than when the dip-samples are poured into water or the bars bored.

The skimmings from the bars are remelted with addition of excess of iron and 10 per cent. of sodium bicarbonate. Clean slag, matte, and bullion are obtained, the bullion separating cleanly from the matte. The quantity of matte obtained in a year is small, amounting to about 500 lb. The composition of the matte before remelting is: Ag, 45.8; Au, 0.2; Cu, 20.9; Fe, 2.8, and Pb, 8.6 per cent.

After remelting with addition of iron the average analysis is: Ag, 11.84; Au, 0.013; Cu, 18.0, and S, 22.4 per cent.

Due chiefly to the fact that no acid is used in treating the precipitate, the slag formed is not corrosive. Crucibles are used for 20 or 25 melts without relining. Old crucibles are cleaned and ground up. The graphite thus obtained is mixed with solution of sugar till it has the consistency of molding-sand and is used in relining the crucibles. A lining 0.5 in. thick is tamped in cleaned crucibles and then slowly dried. After thorough drying and annealing the relined crucibles are good for 10 more melts. The work is done by the men employed in the melting-room.

A portion of one typical clean-up was carefully sampled and assayed. The bullion, slag, and matte were likewise analyzed. The composition of the precipitate was:

	Per Cent.
Silver,	67.71
Gold,	2.62
Copper,	7.43
Zinc,	4.81
Lead,	2.12
Sulphur (combined),	0.39
SiO_2	6.28
$CaCO_3$	5.54
Fe_2O_3	1.84
Al_2O_3	0.73
$MgCO_3$	0.13
KCN, NaCN,	traces
Organic matter, undetermined.	
Total,	99.60

The precipitate before melting contained 8.03 per cent. of moisture.

The results of melting were:

	Ounces.
Dry weight of precipitate melted,	5,363.9
Ground borax-glass,	1,220
Assay-slag,	450
Sand,	375
Total,	7,408.9

The recovery of the precious metals was as follows:

		Ag. Ounces.	Au. Ounces.
5,363.9 oz. precipitate	67.71 per cent. Ag =	3,631 89	
	2.62 per cent. Au =		140.53
4,097.5 oz. bullion,	87.07 per cent. Ag =	3,566.09	
	3.37 per cent. Au =		138.08
3,156.0 oz. slag,	0.96 per cent. Ag =	30.29	
	0.032 per cent. Au =		1.01
43.4 oz. matte,	19.11 per cent. Ag =	8.29	
	0.059 per cent. Au =		0.025
Total recovery,		3,604.67	139.115
Percentage of recovery,		99.25	99.01

The composition of the slag was:

	Per Cent
Ag	0.96
Au	0.03
CuO	1.52
FeO	2.57
Al_2O_3	5.64
CaO	7.24
MgO	2.13
PbO	7.36
SiO_2	23.40
ZnO	8.51
S	0.23
Na_2O	13.20
Br_2O_3	29.74

The analysis of the matte was:

	Per Cent.
Ag	19.11
Au	0.059
Cu	22.95
Fe	31.64
S	23.40
Zn	1.68

The costs of precipitation and melting, Table XX., are given together, as the work is interdependent. Two men are employed in the work, cleaning the boxes and melting. The zinc is cut in the machine-shop, and the cost of cutting is included in the labor and power items.

TABLE XX.—*Cost of Precipitation and Melting.*

	Per Ton of Ore.	Per Fine Ounce, Gold and Silver.
Labor,	$0.0174	$0.0032
Supervision,	0.0168	0.0030
Zinc,	0.0600	0.0111
Coke,	0.0055	0.0010
Charcoal,	0.0053	0.0009
Borax-glass,	0.0158	0.0029
Crucibles and covers,	0.0044	0.0006
Power,	0.0017	0.0003
Miscellaneous supplies,	0.0030	0.0005
Total,	$0.1299	$0.0235

These figures in Table XX. are the average for a period of seven months, during which time 224,390 fine ounces of gold and silver were precipitated and melted.

The returns from the mill-treatment of both unoxidized and oxidized ores are given in Table XXI.

TABLE XXI.—*Mill-Returns on Unoxidized and Oxidized Ores.*

1907, Month.	Quantity Milled.	Unoxidized Ore.							
		Assay of Heads		Quantity Present.		Quantity Recovered.		Recovery.	
		Gold.	Silver.	Gold	Silver.	Gold.	Silver.	Gold.	Silver.
	Tons.	Oz. per Ton	Oz. per Ton.	Oz.	Oz.	Oz.	Oz.	Per Cent.	Per Cent.
March	4,867	0.27	6.78	1,265.42	32,998.0	1,216.76	28,740.0	96.1	87.0
April	4,540	0.20	6.2	908.00	28,184.7	862.60	24,184.7	95.0	85.9
May	4,956	0.21	5.7	1,040.00	28,249.0	991.20	23,909.5	95.2	84.6
June	4,876	0.21	5.1	1,023.96	24,867.6	975.20	20,978.3	95.2	84.3
July	5,400	0.21	4.1	1,134.00	22,140.0	1,080.00	17,678.3	95.2	79.8
August	5,277	0.17	4.9	897.09	25,857.3	839.20	18,669.0	95.4	72.2
Total	29,916	0.209	5.42	6,268.47	162,259.9	5,964.96	134,159.8	95.1	82.6
With silver at $0.50		$7.03		$210,699.22		$190,375.62		90.3	

1908.		Oxidized Ore.							
February	5,190	0.206	5.43	1,069.14	28,181.7	1,017.24	20,907.1	95.1	74.1
March	5,541	0.255	5.75	1,412.95	31,860.7	1,358.91	24,165.8	96.0	75.5
April	5,939	0.213	5.63	1,265.00	33,436.5	1,181.02	27,553.4	93.3	82.4
May	5,207	0.208	6.22	1,083.05	32,387.5	1,031.01	24,528.0	95.2	75.5
June	5,187	0.240	6.39	1,244.88	33,144.9	1,141.14	25,741.4	91.6	77.6
July	5,873	0.245	6.60	1,438.88	38,761.8	1,321.42	25,926.9	91.6	66.8
Total	32,937	0.228	6.00	7,513.90	197,773.1	7,050.74	148,822.6	93.8	75.2
With silver at $0.50		$7.71		$254,198.86		$220,150.10		86.0	

1909.	Quantity Milled.	Oxidized Ore.					
		Assay of Heads.		Assay of Tails.		Recovery.	
		Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
	Tons.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton.	Oz. per Ton	Per Cent	Per Cent.
March	5,838	0.258	6.60	0.015	1.15	94.02	82.49
April	5,923	0.258	6.73	0.015	1.31	94.14	80.44
May	6,043	0.242	6.81	0.016	1.28	93.38	81.15
June	6,082	0.245	7.01	0.016	1.30	93.38	81.42
Total	23,886	0.251	6.79	0.016	1.26	93.62	81.44
With silver at $0.50		$8.58		$0.96		88.81	

X. Cost of Milling and Cyaniding.

The total cost for the year 1908 was approximately $1.50 per ton. So far during 1909 the cost per ton is lower, largely due to an increased saving in cyanide made by the filters and to better quality of wood and cyanide. Included in the labor, salaries, and miscellaneous items are the charges for work done in the shops for the mill. Half the cost of the assay-office is included in the items given below:

	Per Ton of Ore.
Labor,	$0.3053
Salaries,	0.1265
Wood,	0.3205
Cyanide,	0.3005
Lime,	0.0926
Zinc,	0.0682
Shoes and dies,	0.0591
Lubricant,	0.0143
Borax,	0.0110
Coke and charcoal,	0.0112
Assay-supplies,	0.0047
Crucibles,	0.0041
Fittings,	0.0032
Screens,	0.0013
Miscellaneous,	0.0497
Total,	$1.3722

Improvement in Cyanide Practice.

BY E. GYBBON SPILSBURY, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910.)

The recovery of gold and silver from their ores by means of the cyanide process has been so successful in the last few years that any radical improvement would seem impossible; yet the appliance to which I wish to call attention in this paper is really a radical departure from the methods now in general use.

The most modern and approved of these, known as the all-sliming method, depends for its success on the grinding of the ore so fine that practically 90 per cent. of it will pass through a 200-mesh screen. The slimes thus produced are then agitated and aërated in tanks of various types.

The main object of this treatment is to insure such a thor-

ough admixture of the pulp and solution that every particle of the ore is surrounded by a volume of solution sufficient to insure the dissolving of the whole of the gold- and silver-content. In addition, in order to expedite the action of the cyanide, and to oxidize such elements as would, if left in their active state, become cyanicides, air is blown in under pressure. In the Pachuca tank, this air is the active and sole means of agitation.

The chief objections to all the methods of agitation used heretofore are the expense of operating and maintaining the mechanical devices for keeping the pulp in suspension, and the length of time required to obtain a fairly complete extraction of the values. These objections are inherent to any method of agitation effecting a circulation of the pulp and the solution together, by giving the whole mass a circular movement, as in the low-tank system, or a vertical circulation, as in the Pachuca-tank system. In both methods, the particles of ore are kept traveling in the same direction as the solution, and with very little difference of speed; so that, while the whole mass is in violent motion, the relative positions of the ore-particles to their surrounding medium of suspension change but very slowly, and consequently the length of time required for a given extraction is much greater than would be necessary if the movements of the solution and the pulp were not coincident.

The improvement here described is a purely mechanical one, devised to meet this requirement. It rests simply on the discovery of a method of manufacturing a diaphragm of silica sponge, which, while strong enough to support heavy weights, is so evenly porous throughout that air can be passed through it with practically little resistance, and in which, nevertheless, the pores are so minute that no solid matter, however finely divided, can pass through or even into it.

In practice, this diaphragm is placed in the tank as a false bottom, resting on light channel-iron bars, 4 in. above the real bottom. The plates are either 12 by 12 or 12 by 20 in. in size, and are secured to the channel-iron supports, along the lines of intersection, by 0.25-in. carriage-bolts. When the plates are all laid, oakum is driven into the joints, which are then made completely tight by pouring in liquid cement.

This simple operation completes the whole installation. The charge is now run into the tank, and air is admitted from below, under a pressure of from 2 to 6 lb. only, depending on the depth of the charge. Immediately the whole charge becomes a seething mass of uniform but gentle agitation, in which every particle of ore is in constant motion, as is shown in Fig. 1.

No pressure is exerted on any of the air-particles after they have passed through the pores of the diaphragm. They simply levitate up through the mass by reason of their lower specific gravity. No distinct streams or lines of agitation are perceptible to the eye, but the charge shows a very distinct increase of volume, amounting to more than a foot of height in a 6-ft. tank; and the surface becomes covered with a coating of foam or air-bubbles, the thickness of which depends on the volume of air blown through the diaphragm. The full capacity of the sponge varies from 5 to 5.5 cu. ft. of air per minute per square foot of area at 1 lb. pressure.

Under these conditions the action of the cyanide is very rapid and intense. In 80 per cent. of the runs made, an extraction exceeding 50 per cent. of the combined gold- and silver-values has been obtained within the first hour of agitation; and while, in experimental work, the treatment is usually carried on for 12 hr., 6 hr. will probably be found to be the economical period in general practice.

It is found that the consumption of cyanide per ton of ore treated under this method is much less than in either the Pachuca or the mechanically-agitated tanks. This, I believe, is due chiefly to the briefer exposure of the cyanide to the oxidizing effect of the air, but also to the circumstance that we are able to treat effectively a much thicker pulp than the other methods of agitation will permit. With a proportion of 1.5 of solution to 1 of ore, we can obtain the quickest extraction; but, for facility of charging and discharging tanks, we generally make the mixture 2 to 1.

The accompanying records of work done with this porous diaphragm, on a commercial scale, in one of the large mills of the Guanajuato Development Co., show what remarkable extractions are obtained by this method.

One of the important questions we have had to study in the use of this material was, whether the pores of the sponge would

not sooner or later become filled and choked by the very finest particles of ore, thus impairing the efficiency of the plates.

All our experience hitherto goes to prove that no such stoppage need be feared. The pores are so minute and so irregular in shape that apparently no solid matter can find entrance. In treating certain classes of ore we do find that, after a certain number of hours, the air-pressure begins to increase, by reason of a closing of the surface-pores of the diaphragm; but examination under the microscope has shown that this is due to a gradual deposit of lime carbonate on the surface of the plates, formed at the moment of contact of the air with the lime in solution. The removal of this coating, however, offers no difficulty. It can be done, between charges, either by sweeping the surface with a wire broom or by washing it with a weak solution of hydrochloric acid. In either case, the complete removal of the deposit takes place instantaneously, and the air-pressure drops to the normal.

The credit for the successful application of this porous medium to the cyanide process is due to J. E. Porter, of Syracuse, N. Y., who, having acquired the material for an entirely different purpose, conceived at once its possibilities in the cyanide-field, and, by a long series of careful experiments, developed the many advantages of its present adaptation to that field.

Besides the employment of this silica sponge in the treatment-tanks, its usefulness has been demonstrated in the filtering of the solutions, and as a clarifier. Several types of silica-sponge filters are now building, of which probably the simplest consists of a table, from 25 to 30 ft. long and 8 ft. wide, the top of which is constructed of this porous material, under which is an air-tight pan, connected with an exhaust-pump. The pulp being run over the table, the solution is drawn through, leaving a dry cake of the desired thickness. Wash-water is then flowed over this cake, and it is washed in the usual manner. When the cake is finally dried, the table is tilted to a vertical position, and the cake is blown off by air-pressure. The mineral sponge always contains a certain amount of moisture, which causes a film of water to exude when the air-pressure is turned on; and this film, acting as a lubricant, aids the cake to free itself, so that the separation is immediate.

This filter has a capacity of from 40 to 50 tons of dry pulp

per day. The resulting cake contains less than 23 per cent. of moisture when discharged. The simplicity of construction of this filter, its indestructible filtering-medium, and the absence of the innumerable valves and fittings required by all filters of the leaf-type, will recommend it strongly to mill men generally. I think it will increase the saving of values wherever it is introduced.

Another form of filter is being developed by the Just Process Co., using the silica sponge on the sides of a wheel, as

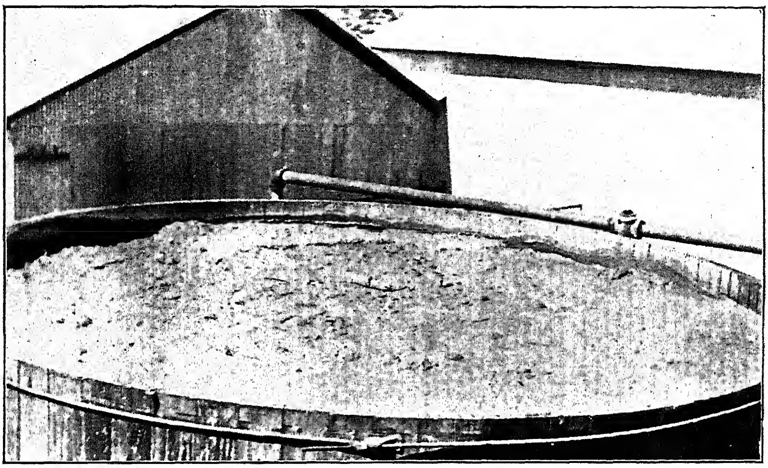

Fig. 1.—Surface of a Tank, 12 ft. Deep, with a Silica-Sponge False Bottom, Showing Agitation of the Charge.

illustrated in Fig. 2. The filter-wheel revolves slowly in a tank containing the pulp, the speed of travel being determined by the porosity of the cake to be formed. The formation of the cake takes place on the lower half of the wheel in submersion, the upper half of the wheel being used for drying the cake. The solution is withdrawn by vacuum through the hub of the wheel. In this case no attempt is made to wash the cake on the filter; instead of this, however, the cake is peeled off, as shown, by a knife on each side, at the end of its travel, and drops down into a cylinder containing rotary beaters revolving at a high speed and reducing the cake again to the consistency of pulp with a minimum amount of wash-water. In practice the product from this repulping-cylinder would

pass to a second wheel and eventually to a third wheel for thorough washing. The filtering-operation is therefore continuous, and the cake as finally discharged is drier than that pro-

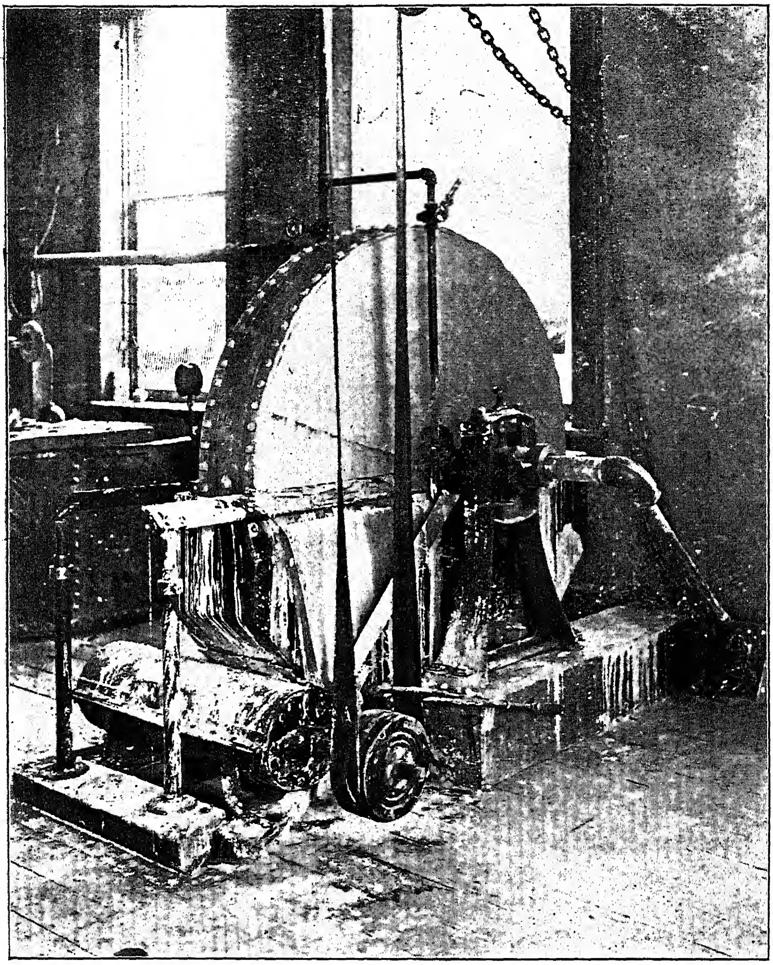

FIG. 2.—ROTARY SILICA-SPONGE FILTER.

duced by any other method. The amount of solution extracted in the first operation is more than 85 per cent., leaving only 15 per cent. of moisture to be replaced by subsequent washing.

An interesting laboratory-method for the rapid preparation of samples of cyanide-slimes, devised by Mr. Porter, consists in the use of a small hollow cylinder of the silica sponge, 4 in. in

diameter by 4 in. in length, closed top and bottom by iron plates. Passing through the top plate is a pipe, which can be attached to any vacuum-line in the mill. The sample of pulp, taken in the usual way, is put into a galvanized-iron receptacle, and the porous cylinder is immersed in it. The vacuum being applied, the water is drawn out through the cylinder, around which a cake forms. When sufficient pulp has thus been solidified, the cylinder is removed to a second bucket and wash-water is drawn through until all the dissolved values have been removed. The cake is then removed with a spatula and dried.

In this way any number of samples can be rapidly prepared, and the long process of washing and decanting in the usual manner is entirely avoided. The resulting sample is also more thoroughly representative of the tank-charge than a sample prepared with the present crude way of washing.

Another application of this silica sponge may be found in the saving of the flue-dust of smelting-operations. Experiments along this line have not yet gone far enough to warrant the publication of detailed plans and results; but it is expected that they will permit the proposal, at an early day, of a method replacing the present "bag-house" system for collecting all the solid particles from flue-gases of every description. This material, composed of nearly pure silica, having a very high fusion-point, and invulnerable to liquid acids or acid vapors, possesses in these respects indisputable advantages over the flannel bags now used.

APPENDIX.

Experimental Tests Made with the Porous Diaphragm at the San Matias Mill, Guanajuato, Mexico, in November and December, 1909.

These tables give weights in tons of 2,000 lb. avoirdupois, and values in Mexican money at $1.33 per gram, or $41.34 per ounce, for gold, and $0.033 per gram, or $1.02 per ounce, for silver.

Test No. 3, November 5 and 6; 12 hours; 2.671 tons "Bolanitos" mine-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 10; and through 200-mesh, 88 per cent. Fresh solution, 5.97 tons. Consumption of cyanide, 232 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent	Per Cent.
Heads.	0	63	3	0.39	0.644	2.17	126	$7.04			
2	2	63	3	0.39	0.960	0.25	83	3.07	88.4	34.2	56 4
3	4	63	4	0.39	0.896	0.17	66	2.42	92.2	47 6	65 7
4	6	63	4	0 39	0.890	0.08	51	1.79	96 4	59 6	74 5
5	8	63	4.5	0.385	0.868	0.17	47	1.77	92.2	62.7	74.9
6	12	63	5	0.385	0.842	0.08	29	1.06	96 4	78.1	85.5

Test No. 4, November 8 and 9; 8 hours; 2.197 tons "El Refugio" dump- and mine-ore. Screen-analysis: on 100-mesh, 3; on 200-mesh, 12; through 200-mesh, 85 per cent. Fresh solution, 4.906 tons. Consumption of cyanide, 228 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	63	4	0.31	0.87	1.83	108	$6.00			
	0.5	63	4	0.305	0.87						
2	1	63	4	0.305	0.84	0.33	77	2.98	82.0	28.7	50.2
3	2	63	4	0.305	0.81	0.17	63	2.30	90.7	41.7	61.7
4	4	63	5	0.300	0.78	0.25	50	1.98	86.4	53.6	67.0
5	6	62	6	0 300	1 12	0.25	38	1.58	86.4	65.9	73.7
6	8	61	6	0 300	1.04	0.17	32	1.28	90.7	70.4	78.7

Test No. 5, November 11 and 12; 22 hours; 2.77 tons "El Refugio" mine-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 11; through 200-mesh, 87 per cent. Fresh solution, 5.9 tons. Consumption of cyanide, 426 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver	Value (Mex)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	88	5	0.280	0.70	1.83	108	$6.00			
2	1	88	5	0.270	0.70	0.33	92 7	3.48	87.3	8.5	42.0
3	2	89	5	0 270	0.75	0 17	72	2.60	90.8	33.4	56.6
4	4	89	5	0.270	1.00	0.08	62	2.15	95.7	42.6	67.5
5	8		5	0 330	0.85	0.08	48	1.68	95.7	55 6	72
6	19	72	6	0.320	0 80	0.08	22	0.82	95.7	79.7	86.4
7	22	89	6	0.320	0.84	0.08	22	0.82	95.7	79.7	86.4

Test No. 6, November 13 and 14; 12 hours; 1.88 tons "Santa Clara" mine-ore. Screen-analysis: on 100-mesh, 3; on 200-mesh, 10; through 200-mesh, 87 per cent. Fresh solution, 5.64 tons. Consumption of cyanide, 450 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution		Assays.			Extractions.		
				KCN	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg	Grams Per Ton	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	81	6	0.30	0.95	8 00	118	$14 53			
2	1	89	6	0.30	0.98	0.17	57	2.10	97.9	51.7	85 6
3	2	95	5	0.30	0.92	0.08	35	1.25	99	69.9	91.4
4	4	94	6	0 295	0 96	0 08	28	1 02	99	76.3	93
5	6	90	6	0.295	0.96	0.08	23	0 86	99	80.6	94.1
6	8	87	6	0 295	0.94	0 08	21	0 79	99	82 8	94 6
7	12	83	6	0 420	0.92	0.08	18	0 69	99	84 8	95.3

Test No. 7, November 15 and 16; 12 hours; 2.16 tons "Puerto Rico" oxidized surface-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 9; through 200-mesh, 89 per cent. Fresh solution, 4.384 tons. Consumption of cyanide, 234 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	77	6	0.39	0.89	1.67	265	$10.96			
2	2	89	6	0 38	0.89	0.17	127	4.41	89.8	52.0	59.0
3	4	90	6	0.38	0.78	0.08	40	1.42	95.2	84.9	87.0
4	6	89	7	0.38	0.816	0.08	32	1.16	95.2	87.9	89.4
5	8	86	7	0.38	0.80	Tr.	22	0.72	99	91.7	93.4
6	10	84	7	0.38	0.80	Tr.	23	0.76	99	91.3	93.0
7	12	80	7	0 38	0.78	Tr.	18	0.59	99	93.2	94.6

Test No. 8, November 16 and 17; 12 hours; 2.723 tons "Puerto Rico" ore. Screen-analysis: on 100-mesh, 3; on 200-mesh, 12; through 200-mesh, 85 per cent. Fresh solution, 6.562 tons. Consumption of cyanide, 234 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	79	6	0.37	0.84	2.50	263	$12.00			
2	2	86	7	0.37	0.94	1.00	183	7.97	60	30.2	35.3
3	4	91	7	0.37	0 85	0.33	148	5.32	86	43.8	55.7
4	6	88	7	0.36	0 87	0.17	36	1.42	93.2	86.3	88.1
5	8	87	7	0.36	0 84	0 08	35	1.25	96.8	86.7	89.6
6	10	82	7	0.36	0 84	0.08	33	1.19	96.8	87.5	90.2
7	12	78	7	0.36	0.87	0.08	28	0.91	96.8	89.4	92.4

Test No. 9, November 18 and 19; 12 hours; 1.825 tons "El Refugio" oxidized ore. Screen-analysis: on 100-mesh, 3; on 200-mesh, 10; through 200-mesh, 87 per cent. Fresh solution, 5.717 tons. Consumption of cyanide, 297 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	64	3	0.28	1.12	3 83	112	$8.78	87		
2	2	76	4	0.27	1.12	0 50	136	5.14	95 6		
3	4	86	5	0.27	0 95	0 17	79	2.84	99	2 90	67.6
4	6	86	5	0.32	0.96	0 08	36	1.28	99	67 8	85.4
5	8	86	6	0.32	0.95	0 08	33	1.19	99	70 5	86 4
6	10	80	7	0.32	0.96	0 08	26	0 96	99	76.8	89
7	12	78	7	0 32	0 96	0 08	21	0 79	99	81.2	91

Test No. 10, November 19 and 20; 12 hours; 2.378 tons "Pinguico" mill-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 10; through 200-mesh, 88 per cent. Solution, 6.444 tons. Consumption of cyanide, 542 grams per ton of ore.

Sample No.	Agitation.	Temperature	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	68	4	0.34	1 04	7.33	864	$38.35			
2	0			0 34	1.04	2.33	343	14.42	68.2	60.3	62.6
3	2	71	5	0.34	1.00	0.83	183	6.48	95.5	78.8	83.1
4	4	77	6	0.33	1.00	0.25	192	6.66	96.6	77.7	82.6
5	6	80	7	0.33	1.00	0.08	55	1.91	99	93.6	95
6	8	80	7	0.33	0.98	0.08	38	1.35	99	95.6	96.5
7	10	80	7	0.32	0.94	0.08	29	1.06	99	96.6	97.2
8	12	78	8	0.32	0.94	0.08	23	0:86	99	97.3	97.7

Test No. 11, November 22; 12 hours; 3.16 tons "Pinguico" mill-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 9; through 200-mesh, 89 per cent. Solution, 9.23 tons. Consumption of cyanide, 441 grams per ton of ore.

Sample No.	Agitation.	Temperature	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0					7 33	864	$38.26			
2	0	65	5	0.35	0 92	2.00	331	13.58	72.8	61 7	64.6 by Con.
3	2	69	5	0.35	0.92	0.33	124	4.53	95 5	85.7	89.2
3+	2	Spec.	from	bottom of	tank.	0.33	124	4 53	95.5	85.7	89.2
4	4	72	5	0 35	0.88	0.25	75	2.80	96.5	91.3	92.7
5	6	70	6	0.34	0.88	0 17	49	1.85	97.7	94.2	95.2
6	8	67	6	0.34	0.86	0.17	37	1.45	97.7	95.7	96.2
7	10	67	7	0.34	0.84	0.08	33	1.26	99	96.2	96.7
8	12	65	7	0.34	0.84	0 17	25	1.04	97.7	97.2	97.2

Test No. 12, November 23; 12 hours; 2.11 tons "Peregrina" mill-run. Screen-analysis: on 100-mesh, 2; on 200-mesh, 13; through 200-mesh, 85 per cent. Solution, 5.71 tons. Consumption of cyanide, 406 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex)	Gold	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent	Per Cent.	Per Cent.
Heads.	0			0.24	0.98	7.00	126	$11.27			
2	0	68	6	0.24	0.92	2.67	164	8.47	61.9		
3	2	68	7	0.35	0.89	0.42	69	2.83	94	46	74.9
4	4	72	7	0.35	0.88	0.42	50	2.21	94	61	80.4
5	6	71	7	0.35	0.84	0.17	21	0.92	97.6	83.4	91.9
6	8	69	7	0.35	0.86	0.17	22	0.95	97.6	82.6	91.6
7	10	69	8	0.35	0.84	0.17	20	0.89	97.6	83	92.8
8	12	62	8	0.35	0.82	0.17	21	0.92	97.6	83.4	91.9

Test No. 13, November 24; 12 hours; 2.986 tons "Peregrina" mill-run. Screen-analysis: on 100-mesh, 2; on 200-mesh, 12; through 200-mesh, 86 per cent. Solution, 6.09 tons. Consumption of cyanide, 408 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	67	5	0.38	0.90	7.00	126	$11.27			
2	2	72	7	0.37	0.90	0.50	50	2.32	92.9	61	79.3
3	4	72	7	0.37	0.86	0.33	28	1.36	95.3	77.8	88.0
4	6	70	7	0.37	0.86	0.33	20	1.10	95.3	84.2	90.2
5	8	68	7	0.365	0.86	0.33	19	1.07	95.3	85	90.5
6	10	65	8	0.365	0.82	0.17	15	0.72	97.6	88.1	93.7
7	12	65	8	0.365	0.78	0.17	13	0.66	97.6	89.7	94.2

Test No. 14, November 25; 14 hours; 1.61 tons "Peregrina" mill-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 10; through 200-mesh, 88 per cent. Solution, 4.358 tons. Consumption of cyanide, 271 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	66	5	0.24	0.81	6.5	169	$14.22			
2	2	68	6	0.24	1.00	0.67	55	2.70	89.7	67.5	81
3	4	78	6	0.36	0.86	0.33	24	1.23	95	85.8	91.4
4	6	77	6	0.36	0.86	0.17	13	0.65	97.4	92.3	95.4
5	8	74	6	0.36	0.86	0.17	15	0.72	97.4	91.2	95
6	10	70	6	0.36	0.86	0.17	13	0.65	97.4	92.3	91.4
7	12	67	6	0.36	0.86	0.25	12	0.72	96.2	92.9	95
8	14	64	7	0.36	0.84	0.17	11	0.58	97.4	93.6	96

Test No. 15, November 27; 12 hours; 1.75 tons "La Luz" mine- and dump-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 10; through 200-mesh, 88 per cent. Solution, 5.55 tons. Consumption of cyanide, 317 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution		Assays.			Extractions		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr	Deg F.	Lb	Per Cent.	Kg.	Grams Per Ton	Grams Per Ton.		Per Cent	Per Cent.	Per Cent.
Heads.	0	70	5	0.21	1.00	3.33	142	$9.21			
2	2	70	5.5	0.37	0.98	0.33	48	2.01	90	66.2	78.3
3	4	68	6.5	0.37	0.98	0.25	47	1.88	92.5	66.9	79.6
4	6	68	8	0.37	0.98	0.17	41	1.57	94.3	71.2	83
5	8	68	8	0.37	0.98	0.17	24	0.95	94.3	83.9	89.7
6	10	67	5	0.37	0.98	0.17	21	0.91	94.3	85.2	90.2
7	12	67	5	0.37	0.98	0.17	18	0.81	94.3	87.4	91.2

Test No. 16, November 29; 12 hours; 1.625 tons "La Luz" mine- and dump-ore. Screen-analysis: on 100-mesh, 2; on 200-mesh, 10; through 200-mesh, 88 per cent. Solution, 5.29 tons.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent
Heads.						3.33	142	$9.21			
2	1	65	5	0.24	1.06	2.33	145	7.88	30		14.5
3	2	65	5	0.36	0.90	0.42	56	2.41	87.4	60.9	73.9
4	4	65	8	0.355	0.86	0.17	39	1.50	94.9	72.5	83.7
5	6	64	8	0.355	0.86	0.17	37	1.43	94.9	73.3	84.5
6	8	64	8	0.355	0.86	0.08	26	0.93	97.6	81.7	85.8
7	10	63	8	0.355	0.84	0.08	19	0.73	97.6	86.8	92.1
8	12	62	8	0.355	0.80	0.08	24	0.90	97.6	83.2	90.3

Test No. 21, December 5; 12 hours; 1.41 tons "Guanajuato mixture." Screen-analysis: on 100-mesh, 2; on 200-mesh, 12; through 200-mesh, 86 per cent. Solution, 4.244 tons. Consumption of cyanide, 1,200 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	64	3	0.145	0.68	3.42	294	$14.24			
2	After concentration.					1.83	203	9.12	46.5	31.00	36.00
3	2	63	4	0.28	0.62	0.67	108	4.45	80.4	63.3	68.8
4	4	63	5	0.28	0.52	0.42	67	2.77	87.7	77.2	80.6
5	6	63	5	0.27	0.48	0.17	27	1.11	95.1	90.8	92.2
6	8	62	5	0.27	0.44	0.17	31	1.24	95.1	89.5	91.3
7	10	62	5	0.27	0.36	0.17	25	1.04	95.1	91.6	92.7
8	12	62	6	0.26	0.34	0.17	25	1.04	95.1	91.6	92.7

Test No. 22, December 6 ; 12 hours ; 2.677 tons "Guanajuato mixture." Screen-analysis: on 100-mesh, —; on 200 mesh, —; through 200-mesh, 84 per cent. Solution, 6.367 tons. Consumption of cyanide, 714 grams per ton of ore

Sample No.	Agitation	Temperature	Air-Pressure	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg F.	Lb.	Per Cent.	Kg	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	66	4	0.275	0.96	3.42	294	$14.24			
2	2	65	7	0.275	0.96	0.83	149	7.12	75.7	34.0	50.0
3	4	65	7	0.275	0.94	0.33	86	3.27	90.3	70.8	77.1
4	6	65	7	0.275	0.96	0.25	78	2.90	92.7	74.2	79.6
5	8	64	8	0.27	0.96	0.17	72	2.61	95.0	75.6	81.6
6	10	64	8	0.27	0.98	0.17	68	2.46	95.0	76.9	82.7
7	12	64	8	0.27	0.98	0.17	66	2.40	95.0	77.6	83.2

Test No. 24, December 10 ; 22 hours ; 3.878 tons "La Tula" ore, from La Luz district. Screen-analysis : on 100-mesh, 4 ; on 200 mesh, 12 ; through 200-mesh, 84 per cent. Solution, 6.327 tons. Consumption of cyanide, 978 grams per ton of ore.

Sample No.	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg F	Lb.	Per Cent.	Kg	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent.	Per Cent.
Heads.	0	60				21.28	1,710	$84.73			
2	1	60	2	0.335	1.00	7.5	1,026	43.82	64.8	40.00	48.3
3	2	60	3	0.335	1.00	4.5	926	36.54	78.9	45.9	56.9
4	4	60	3	0.305	1.00	1.83	679	24.84	91.4	60.3	70.7
5	6	60	4	0.300	1.00	1.67	498	18.65	92.2	71.1	78.0
6	8	60	4	0.35	0.98	1.33	401	15.00	93.8	76.6	82.3
7	10	60	4	0.35	0.98	1.00	253	9.67	95.3	85.1	88.6
8	12	60	5	0.34	0.96	1.17	231	9.17	94.6	86.5	89.2
9	14	59	5	0.34	0.96	1.00	203	8.03	95.3	88.2	90.5
10	16	59	5	0.34	0.98	1.00	198	7.86	95.3	88.5	90.7
11	18	59	5	0.33	0.98	1.00	184	7.40	95.3	89.7	91.2
12	20	59	6	0.33	0.98	0.83	197	7.60	96.1	88.5	91.0
13	22	58	6	0.33	0.98	1.00	165	6.87	95.3	90.4	91.9

Test No. 25, December 11 ; 16 hours ; 1.909 tons "La Tula" ore. Screen-analysis : on 100-mesh, 4 ; on 200-mesh, 13 ; through 200-mesh, 83 per cent. Solution, 395 tons. Consumption of cyanide, 800 grams per ton of ore.

Sample No	Agitation.	Temperature.	Air-Pressure.	Titrations Per Ton of Solution.		Assays.			Extractions.		
				KCN.	CaO.	Gold.	Silver.	Value (Mex.)	Gold.	Silver.	Values.
	Hr.	Deg. F.	Lb.	Per Cent.	Kg.	Grams Per Ton.	Grams Per Ton.		Per Cent.	Per Cent	Per Cent.
Heads.	0	59	2	0.38	1.00	14.07	368	$30.85			
2	2	59	3	0.37	1.00	2.08	403	16.06	82.5		
3	4	59	4	0.36	0.98	1.17	271	10.50	61.7	26.4	47.9
4	6	59	5	0.36	0.98	0.67	187	6.06	95.6	49.2	80.4
5	8	58	5	0.35	0.98	0.58	155	5.88	95.9	57.9	80.9
6	10	58	5	0.35	0.98	0.58	131	5.09	95.9	64.4	83.5
7	12	58	6	0.35	0.96	0.58	121	4.76	95.9	67.2	84.6
8	14	58	6	0.34	0.96	0.58	118	4.66	95.9	67.9	84.9
9	16	57	6	0.34	0.96	0.50	116	4.48	96.5	70.4	85.5

The Chemical Control of Slimes.*

BY HARRISON EVERETT ASHLEY, PITTSBURG, PA.

(Pittsburg Meeting, March, 1910.)

Slimes are usually defined as all material passing a certain sized sieve, which is invariably the finest sieve employed by each metallurgist in his tests; 100-mesh and 200-mesh have been taken as the limits by different writers in recent volumes of the *Transactions*; 150-mesh is common practice.

The reader of the first two volumes of Richards[1] is impressed by the inflexibility with which all grinding-arrangements are praised or condemned according as they make little or much slime. Until recently, one of the principal aims of the mill-man was to keep down the slimes. He was obliged to have separate apparatus for sand- and slime-treatment. The latter was sometimes puzzling and erratic. Often it was found most economical to omit treatment of the slimes entirely, or to store them in immense basins or heaps, waiting for possible future improvements in methods of treatment.

Recently, improved processes of agitation and filtration have made possible the successful extraction of values from many slimes. The tendency, when the metal is exceedingly finely disseminated, is to cut out all sand-treatment and to slime everything.

These processes have been almost entirely mechanical in their conception, and it is the object of the present paper to show that certain chemical considerations may greatly hinder or assist their successful operation.

It has long been a source of dissatisfaction to chemists that their analyses fail to be of much help to the clay-worker; that it is a matter of common occurrence for clays of closely identical ultimate chemical constitution to show the widest possible divergence in physical properties.

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[1] *Ore Dressing* (1903).

In 1901 I had the pleasure of contributing to the Institute[2] a paper that established the nature of igneous fusions. I am pleased again to be able to present a fundamental investigation in silicate chemistry, this time on the nature of slimes.

The separation by Schloessing, in 1874, of kaolin into a crystalline portion and a colloidal portion was a step long ignored by most chemists.

By the term "colloidal" is meant anything not crystalline. It includes everything covered by the term amorphous. In clays and soils the crystal grains are enveloped with a coating of gelatinous colloid matter. The same is true of slimes.

Many of the problems presented to the Clay Products Section of the Technologic Branch of the U. S. Geological Survey seemed so intimately connected with the chemical nature of the colloid matter in clays that the first investigation begun was upon the chemical reactions of the colloid matter in clays.

Among these questions are the control of the plasticity of clays, the control of the casting-process for making clay-wares, the securing of maximum density and strength in clay-wares, the analytical separation of colloid and crystal matter in clays, the determination of the sizes of the grains of clay down to 0.01 mm., the prevention of excessive loss from the cracking of clay-wares in drying, the prevention of white scum on brick, etc.

The investigation was continued throughout the year 1909, and is now practically finished. It is intended to work during the coming year on the technical application of the principles discovered.

As some of the metallurgical slimes are true clays, and as the colloidal silica from crushed quartz rock is identical with a portion of the colloidal matter encountered in clay-working, and the colloidal matter from feldspar with another portion, some principles of colloid chemistry common to both industries will no doubt interest those metallurgists dealing with the treatment of slimes or with wet-concentration methods.

It is customary to call the colloid a *gel* in its solid condition, and a *sol* when in solution or suspension. The only difference between the two terms is one of physical state.

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[2] *Trans.*, xxxi., 855 (1901).

In Fig. 1, the tubes shown have each 5 g. of clay, and various additions of sodium hydroxide have been made, the volume made up to about 50 cc., the tubes shaken 1 hr. on a rotary shaker, and allowed to settle about 24 hr. The tube with 8 cc. of $\frac{N}{10}$ NaOH shows about 3 cc. of crystal sediment and 47 cc. of solution containing clay in the *sol* form. The tubes with 0, 4, and 8 cc. of 2.5 normal NaOH show a sediment consisting of a mixture of crystal grains and colloid *gel*—the ordinary appearance of clay. The object of the dialysis was to eliminate soluble salts and their effects.

The work during the year shows a very striking parallelism between the chemical behavior of the colloid matter of the clays and the chemical reactions of fatty acids and soaps.

In each case a large acid radical is somewhat loosely bound to a positive element. The colloid acid radicals of clays are very likely polymers of the simple compounds.

The free acids in both cases are insoluble in water, as are also the compounds with bivalent or trivalent metals, such as lime, magnesium, iron, and aluminum compounds.

Table I. illustrates the chemical reactions of clay or slime colloids:

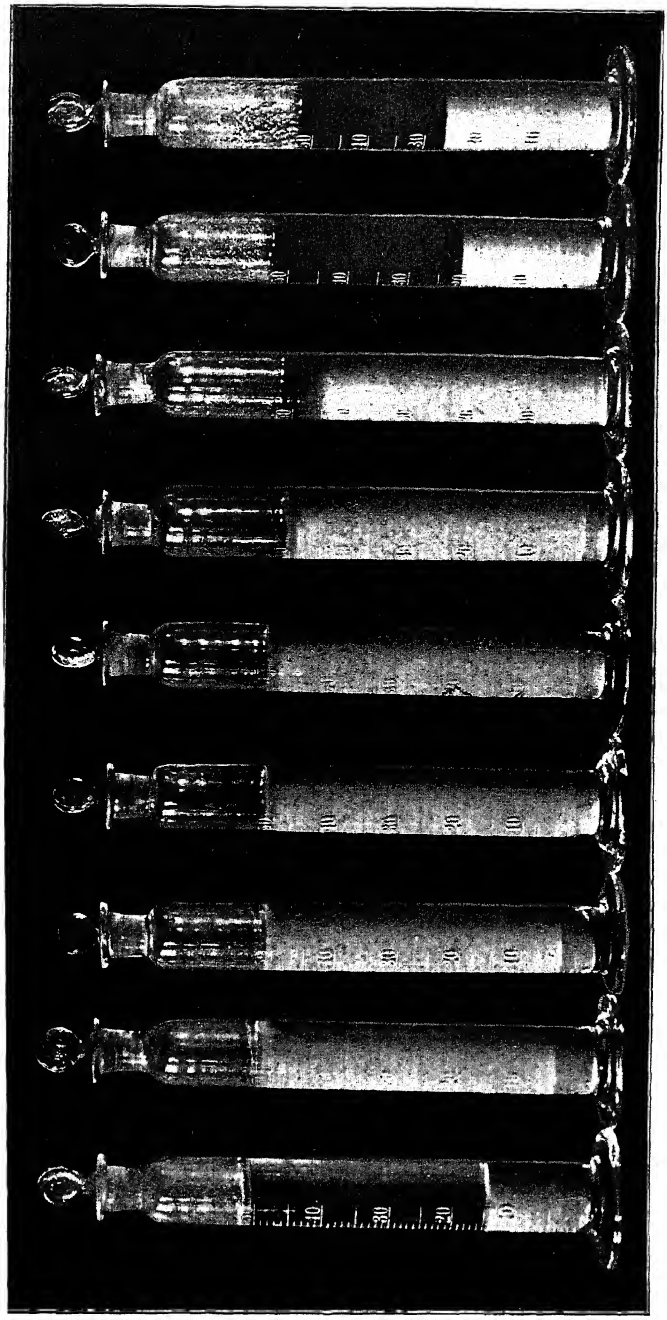
TABLE I.—*Reactions of Clay or Slime Colloids.*

K *sol*	+	KOH (excess)	=	K *gel*	+	KOH (excess)	Salting-out.
K *sol*	+	KCl (excess)	=	K *gel*	+	KCl (excess)	Salting-out.
Na *sol*	+	HCl	=	H *gel*	+	NaCl	Coagulation.
Na_2 *sol*	+	$CaSO_4$	=	Ca *gel*	+	Na_2SO_4	Coagulation.
H *gel*	+	NaOH	=	Na *sol*	+	H_2O	Deflocculation.
Ca *gel*	+	$Na_2C_2O_4$	=	Na_2 *sol*	+	CaC_2O_4	Deflocculation.

If we were to substitute the word "oleate" or the symbol $C_{18}H_{33}O_2$ for *gel* and *sol* in the above equations, they would represent the behavior of one of the commonest fatty acids.

Since it has been demonstrated by W. D. Richardson[3] that transparent soaps are wholly colloidal, and opaque soaps are partly or wholly crystalline, it seems but natural that the two classes of colloidal substances, though mineral and organic respectively, should behave similarly. This is emphasized by the fact that certain highly colloidal clays are in practice added to soaps.

[3] *Journal of the American Chemical Society,* vol. xxx., pp. 414 to 420 (1908).



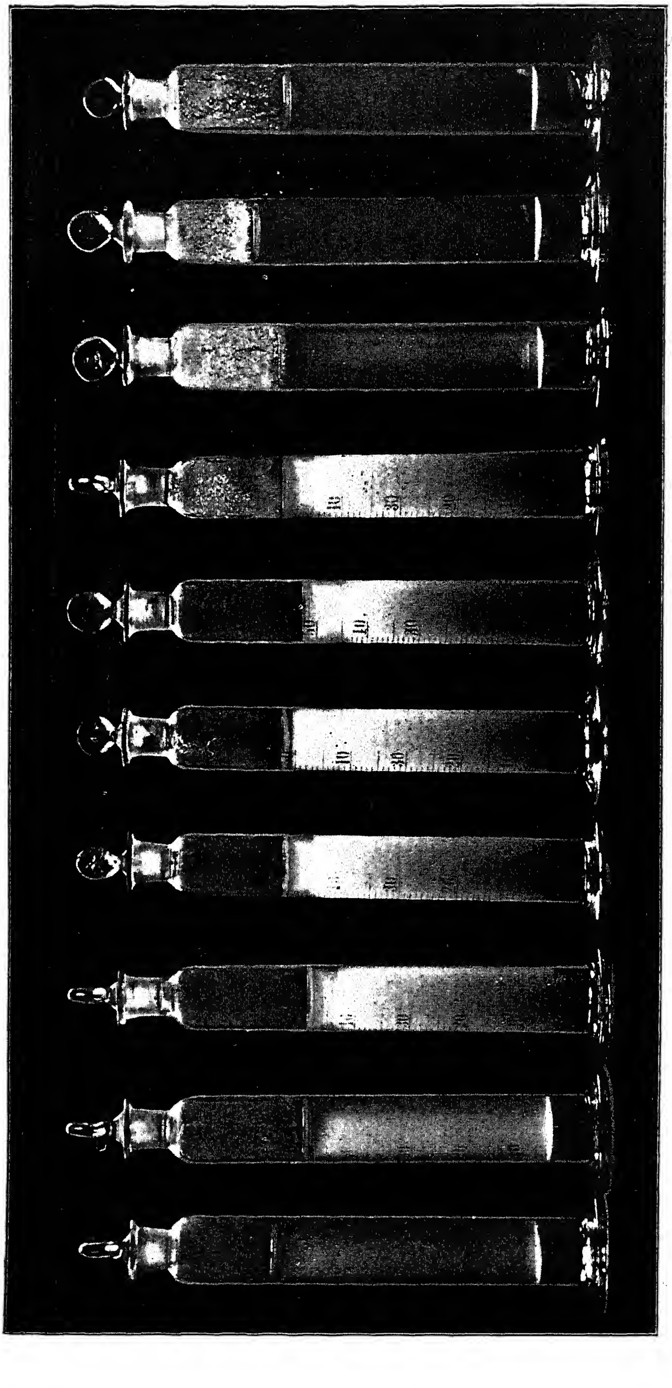
0 1 2 8 10 16 48

Tenth Normal NaOH, cc.

4 8

2.5 Normal NaOH, cc.

FIG. 1.—THE ACTION OF SODIUM HYDROXIDE UPON A CLAY FROM WHICH THE SOLUBLE SALTS HAVE BEEN REMOVED BY DIALYSIS.



1 0.5 0.2 0.1 0 0.5 20 50 5 20

Tenth Normal HCl, cc. Tenth Normal NaOH, cc. 2.5 Normal NaOH, cc.

FIG. 8.—THE ACTION OF HYDROCHLORIC ACID AND SODIUM HYDROXIDE UPON COMMERCIAL DRY-GROUND QUARTZ (SAND ROCK).

The alkali compounds are in both cases soluble or go into suspension in sufficient excess of pure water (tubes 1 to 16, Fig. 1), but are salted out by excess of alkalies (tubes 48, 4, 8, Fig. 1) or alkali salts. "Salting-out" is a soap-boiling term. Soap is separated from solution in water by adding an excess of common salt; for which the water has a greater affinity than for the soap. The alkali compounds may be redissolved by washing by decantation or on a filter, or by sufficient dilution with pure water.

Acids or bivalent or trivalent salts added to the solutions or suspensions of the alkali compounds effect coagulation by

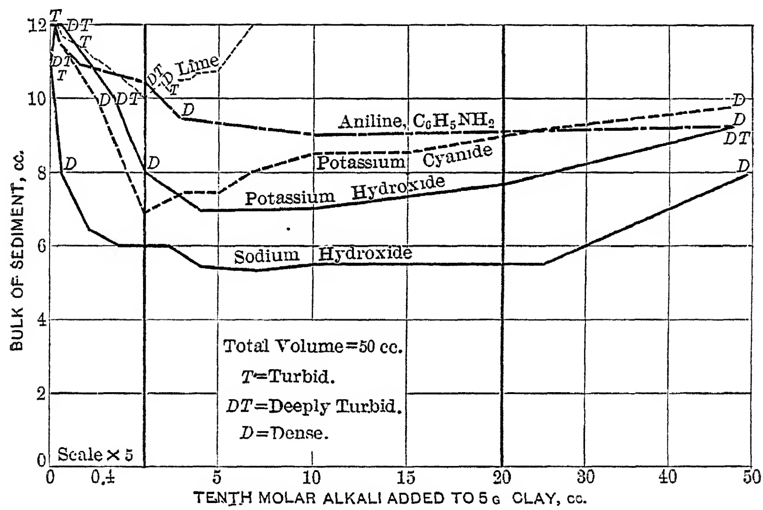


FIG. 2.—COMPARISON OF EFFECT OF ALKALIES ON KAOLIN (HARRIS CLAY CO., PENLAND, N. C.).

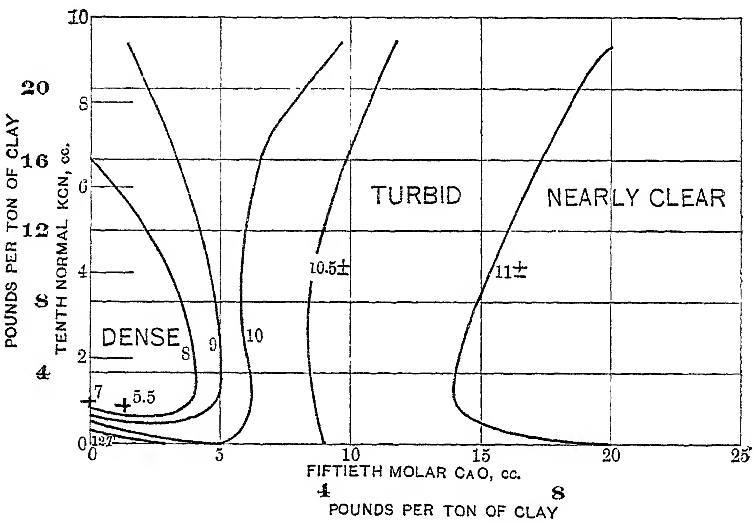
double decomposition. Conversely, the addition of alkalies to the acid colloid effects formation of the alkali compound. Also the addition of alkali salts, the acid radicals of which form insoluble compounds with the bases in the colloid compound or soap, results in the formation of the alkali compound of the colloid or fatty acid.

These statements are represented by the equations of Table I.

In the present paper I use the words "solution" and "suspension" interchangeably. While in ordinary solutions the forces that keep the molecules in solution far overbalance that of gravitation, they are probably only just sufficient

in the case of the larger clay particles to retard the fall of the immense polymerized clay molecules; so that one sample of clay settles clear in 30 min., another in 24 hr., another in 6 months, and finally sufficiently large forces and sufficiently small molecules are reached so that the particles are in permanent solution. The clays and slimes are at the boundary where solution and suspension are identical or overlap.

The effects of several alkalies upon a sample of clay are shown in Fig. 2. In each case 48 cc. of the solution and 5 g. of clay were put into a 50-cc. graduated tube, shaken gently for 1 hr., and allowed to stand over night. The bulk volume



Figures on curves give volume of sediment from 5 g. of clay.

FIG. 3.—SOLUTION OF KAOLIN BY POTASSIUM CYANIDE AND LIME.

occupied by the sediment was measured, and the character of the liquid noted as clear, *C;* nearly clear, *NC;* turbid, *T;* deeply turbid, *DT* (objects not visible); or dense, *D* (light not transmitted). A certain small addition of lime acts to put the clay colloid into suspension, a larger amount to coagulate it. Potassium cyanide acts much like an alkali.

All solutions used in this investigation were made up approximately tenth molar, except where specially noted; *i. e.*, the sum of the atomic weights of the elements given by the chemical formula was taken and divided by 10, to give the weight in

grams of the substance that should be dissolved in water to make 1,000 cc. of solution. A few examples are given below:

Reagent.	Formula	Strength.	Preparation
Sodium hydroxide,.... .	$NaOH$[a]	0 1 molar	4 g diluted to 1 liter.
Sodium hydroxide,.... .	$NaOH$	2.5 molar	100 g. diluted to 1 liter.
Potassium hydroxide, . .	KOH	0 1 molar	5.611 g. diluted to 1 liter
Potassium cyanide,.... . .	KCN	0 1 molar	6.511 g. diluted to 1 liter.
Lime..	CaO	0.02 molar	Slaked in boiling water, filtered Amount in solution determined. Diluted to give fiftieth molar.
Calcium sulphate,...	$CaSO_4$	0.02 molar	Plaster of paris slaked in water, filtered. Amount in solution determined. Diluted to give fiftieth molar.
Calcium chloride (fused),. ...	$CaCl_2$	0.1 molar	11.1 g. per 1 liter.
Magnesium chloride (fused),	$MgCl_2$	0.1 molar	9.524 g. per 1 liter.
Magnesium sulphate,.....	$MgSO_4\ 7\ H_2O$	0.1 molar	24.653 g. per 1 liter.

[a] The sum of the atomic weights in this formula is $23 + 16 + 1 = 40$.

Fig. 3 shows the joint effect of cyanide and lime on the same clay. About 2 lb. of cyanide per ton has a maximum effect, reducing the bulk of the sediment from 12 cc. to 7 cc. Addition of about 0.7 lb. of lime still further reduces the sediment to 5.5 cc. This sediment is entirely crystalline, all of the colloid matter being in suspension. If the lime is increased to 6.5 lb. per ton, the liquid becomes nearly clear by recoagulation of the colloid matter.

It has been shown independently during the past year by Comstock[4] and by me[5] that the rate of settling of a slime, so far as temperature is concerned, varies inversely as the viscosity of the water.

A slime that is partly in the *sol* state neither settles nor filters well. It must be fully coagulated. If an excess of the coagulant is used, the viscosity of the solution is raised and the rate of settling retarded.

To keep down the viscosity, the most effective coagulant possible should be employed, so as to keep down the amount of dissolved substances in the liquid. Fig. 4 shows the comparative effects of lime (calcium oxide, CaO) and calcium chloride ($CaCl_2$, a very cheap chemical) upon a kaolin to which potassium cyanide has been added. Three cc. of tenth molar CaO

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[4] *Electrochemical and Metallurgical Industry*, vol. vii., No. 2, p. 74 (Feb., 1909).

[5] *Mining and Scientific Press*, vol. xcviii., No. 24, p. 831 (June 12, 1909).

is required to effect the same coagulation as 1 cc. of tenth molar $CaCl_2$, that is, in this case, calcium as the chloride is three times as effective as calcium as the oxide. (Owing to the impossibility of preparing tenth molar lime, the equivalent amount of fiftieth molar lime was actually employed).

Probably gypsum (hydrated calcium sulphate, $CaSO_4.2H_2O$), magnesium chloride ($MgCl_2$), or magnesium sulphate ($MgSO_4$) would be about as effective.

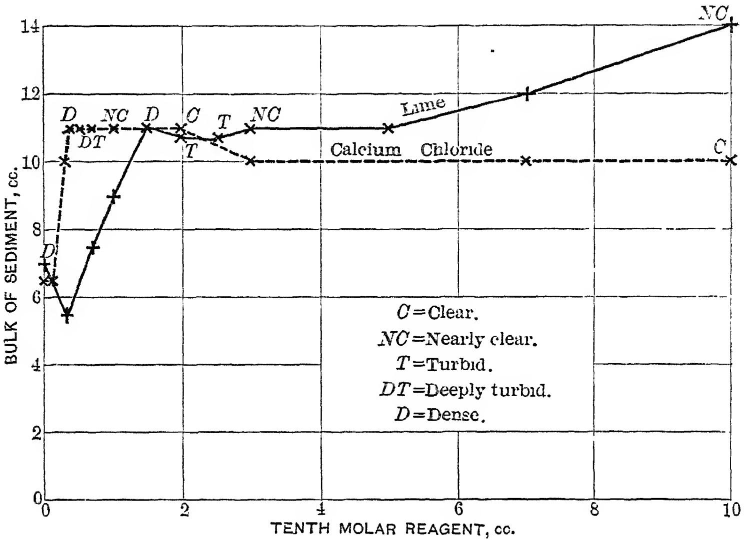


FIG. 4.—COMPARISON OF LIME AND CALCIUM CHLORIDE AS CLARIFIERS OF A SOLUTION CONTAINING 5 G. OF KAOLIN AND 1 CC. OF TENTH MOLAR POTASSIUM CYANIDE.

I believe it to be a mistake with a slime of this clayey character to use more lime than that amount just sufficient to prevent the escape of free hydrocyanic acid (HCN) gas, just sufficient to overcome acidity, and that solutions should be cleared by one of these cheap salts.

Figs. 5 and 6 show the effect of these reagents on the clarification and bulk of sediment of orthoclase feldspar and quartz.

The action of an alkali in dissolving the colloid matter from the surface of the crystal grains breaks up the floccules, or lumps, and is called "deflocculation" by agricultural chemists.

The manner in which a salt of a bivalent metal suppresses the

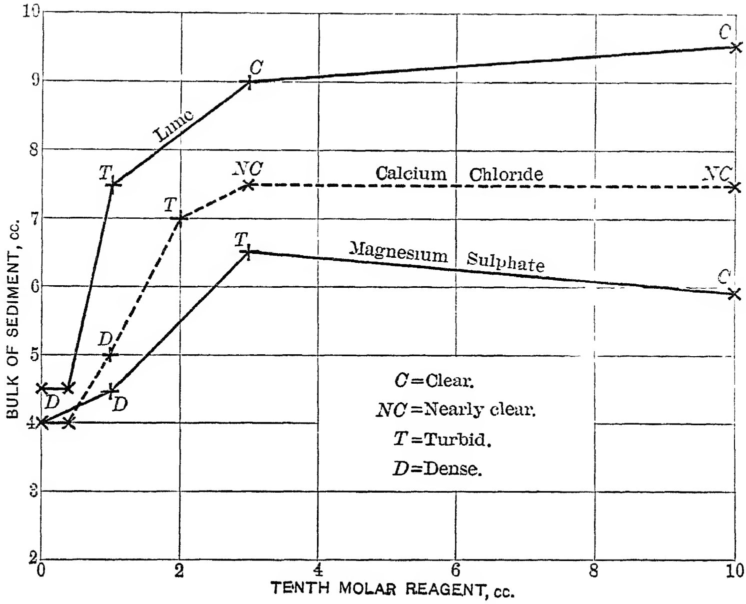


FIG. 5.—CLARIFICATION OF ORTHOCLASE (5 G. IN 98 CC. OF LIQUID) BY LIME, CALCIUM CHLORIDE, AND MAGNESIUM SULPHATE.

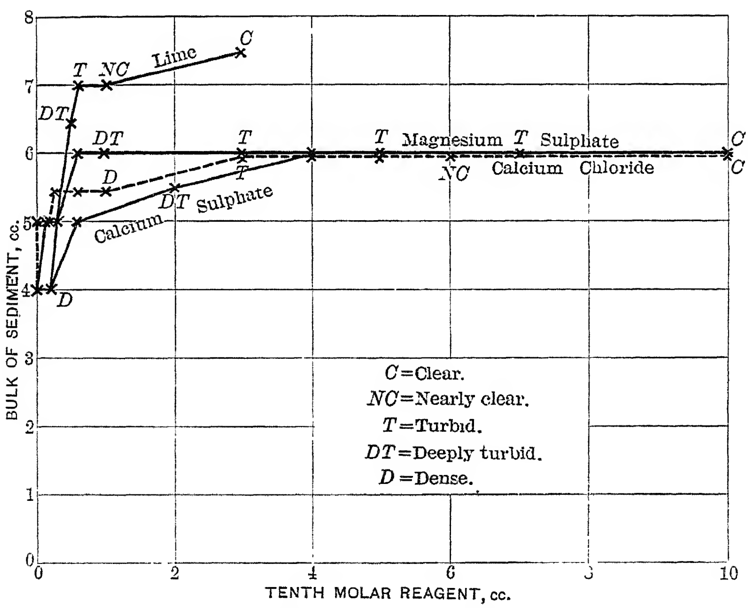
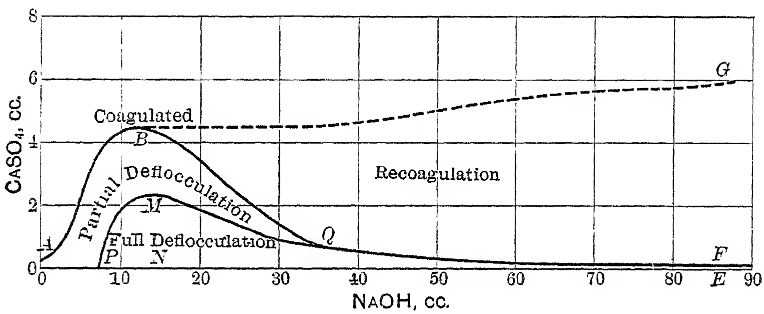


FIG. 6.—CLARIFICATION OF QUARTZ (5 G. IN 98 CC. OF LIQUID) BY LIME, MAGNESIUM SULPHATE, CALCIUM CHLORIDE, AND CALCIUM SULPHATE.

deflocculating effect of an alkali is shown in Fig. 7. The amounts of additions of sodium hydroxide are shown by the abscissas. Thus with no addition to 5 g. of this clay, it was partially deflocculated; with about 7 cc. of $\frac{N}{10}$ NaOH (point *P*) the clay was fully deflocculated; and further additions up to a total of 85 cc. of $\frac{N}{10}$ NaOH (point *E*) continued to keep all the colloid clay in suppression. The amounts of additions of calcium sulphate are shown by the ordinates. About 0.2 cc. of tenth molar $CaSO_4$ (*A*) sufficed completely to coagulate the clay, with complete clarification of the liquid; and further additions up to a total of 8 cc. of tenth molar $CaSO_4$ continued to keep the clay wholly coagulated. When both calcium sul-



5 g. of Tennessee ball clay No. 3 (Mandle-Sant Clay Co.); tenth molar calcium sulphate; tenth normal sodium hydroxide.

FIG. 7.—THE OPPOSITION OF CALCIUM SULPHATE TO THE DEFLOCCULATING-ACTION OF SODIUM HYDROXIDE.

phate and sodium hydroxide were added to the clay, the colloid matter was wholly dissolved only with those quantities represented by points in the area *P M Q F E P*. In other words, with this clay, slight additions of $CaSO_4$ very rapidly narrowed the range of concentrations of NaOH effecting deflocculation until 2 cc. tenth molar $CaSO_4$ limited full deflocculation to but one proportion of $\frac{N}{10}$ NaOH, *viz.*, 15 cc. at point *M*. Concentrations of calcium sulphate given by the line *A B G* kept the clay wholly coagulated, and at nearly the original bulk. Larger amounts of calcium sulphate also kept the clay coagulated, but the bulk become gradually larger as the distance above the line *A B G* increased. In the area

Q B G F, the clay was momentarily deflocculated as if by only 15 cc. $\frac{N}{10}$ NaOH when the sodium hydroxide was first added; but when thorough mixture and full reaction was effected the clay was recoagulated. The bulk of the recoagulated clay was very large along the line *Q F*, and gradually decreased to the line *B G*, where it had nearly the original bulk. The area *A B Q M P A* represents proportions of calcium sulphate and sodium hydroxide with which the clay assumed an intermediate condition, partly coagulated and partly deflocculated.

If one desires to settle out the granular matter, the most favorable position on the diagram is where the least amount of alkali effects full deflocculation. This principle is capable of extensive employment in ore-washing, and wherever perfect classification is desired, as in preparation for jigs, shaking-tables, canvas tables, etc. The colloid matter, being dissolved in water, does not mechanically hinder the fall of the grains. Wherever the clayey gangue is in a suitable colloid condition, it may be dissolved away by a suitable small amount of alkali. This idea might be of assistance in the hydraulic mining of clay-bearing gravels. It undoubtedly may be used where sand is to be washed clean for building purposes, or wherever clean separate grains are desired.

On the other hand, if one wishes to transport mixed colloid and granular matter in a launder, the grains will be carried by the least amount of water, or least slope, when the condition of the mixture is represented by a position in the area marked "recoagulation."

It is even possible that these principles, if not too expensive, can replace dredging to some extent in keeping open the channels of rivers that are clogged by fine clay.

Pure water has a decided solvent effect upon crystalline minerals, as is illustrated by Table II.

TABLE II.—*Solvent Effect of Water on Quartz.*

Water, cc., .	80	80	80	80	80	80	80	80	80	80	80	80
Quartz, g., .	5	10	15	20	30	40	50	80	100	120	160	200
Clear, . . .	...	...	...	...	...	...	...	...	.....	.....	.....	18
Nearly clear, .	...	...	...	...	...	...	...	...	.....	.....	6	.....
Turbid, . . .	...	8	6	6	4	3	5	4	7	14	14	.....
Deeply turbid,	79	67	67	65	67	63	52	41	28	14	.....	.....
Dense, . . .	...	...	...	...	...	...	...	...	.....	.....	.....	.....
Sediment, . .	4	9	13	17	21	29	43	65	80.5	102	118	138

The limit of solubility in this case is reached when the solution becomes deeply turbid, as with 5 g. of quartz to 80 cc. of water. When the proportion reaches 160 g. of quartz to 80 cc. of water, the impurities in the quartz suffice to start salting-out or coagulation of the silica *sol.*

Figs. 8, 9, and 10 show the action of hydrochloric acid, as well as sodium hydroxide, upon quartz, mica, and feldspar. The resulting solutions or suspensions are largely of colloidal nature in these cases. The action is almost instantaneous. Therefore, to keep down slimes in wet grinding, the volume of water should be kept low.

I have found the solvent effect of ordinary running water on silica to increase very rapidly with temperature in the ordinary range of atmospheric temperature. Therefore, to keep down slimes, water as cold as practicable should be used in grinding.

It is well known that the rate of solution is proportional to the surface exposed, and hence is greater the more finely a mineral is ground.

According to Caetani and Burt,[6]

> "All the experiments show that there is some loss in cyanide every time a solution comes in contact with the ore, even when all the values have already been brought into solution ; and the loss in cyanide is proportional to the strength of the solution used."

This seems to indicate a solvent action by the alkaline liquid beyond that due to water alone. Michaelis[7] has shown such an action of lime on silica.

In those cases where all-sliming has proved a success, that success has rested upon the fact that a sufficient proportion of crystal mineral grains was introduced by the fine grinding to keep the mixture of crystals and colloids sufficiently open and porous for the percolation of liquids. The more completely the colloid portion of the slimes is suppressed, the more rapidly and successfully will these processes proceed, so long as the filter-press cake will hold together.

I have a suggestion to make here, which is, that the formation of the colloid matter can probably be suppressed to a considerable extent by grinding in the presence of such soluble compounds as will tend to keep the colloid matter coagulated

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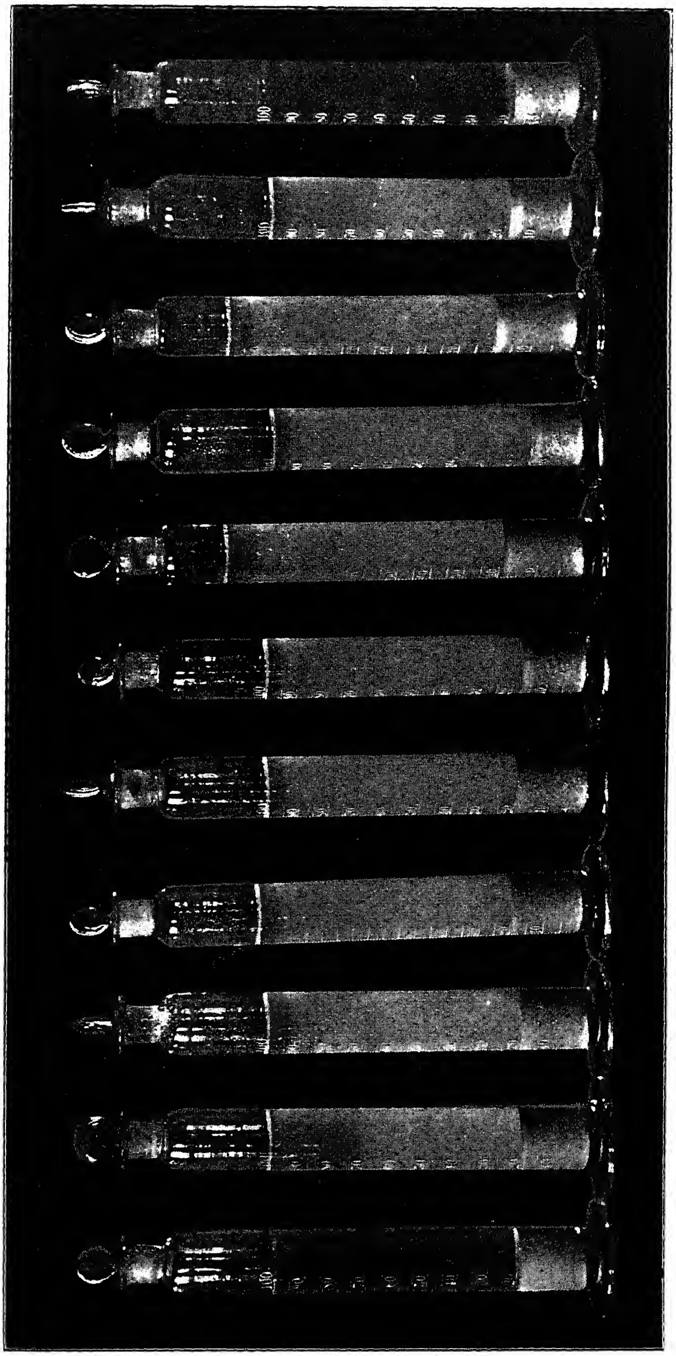
[6] *Trans.*, xxxvii., 29 (1907).

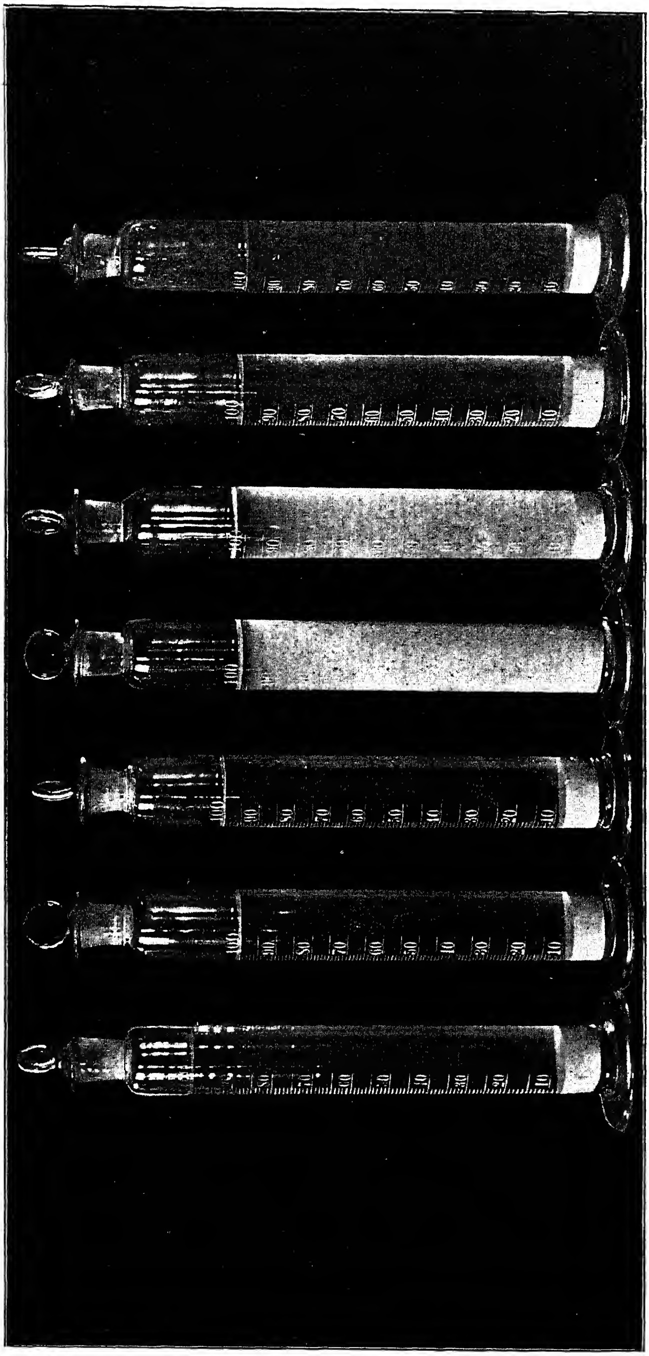
[7] *Tonindustrie-Zeitung*, vol. xxxiii., No. 114, pp. 1243 to 1245 (Sept. 28, 1909).



1 0.3 0.1 0 0.1 0.3 1 3 10 30 100

Tenth Normal HCL, cc. Tenth Normal NaOH, cc.

FIG. 9.—THE ACTION OF HYDROCHLORIC ACID AND SODIUM HYDROXIDE UPON COMMERCIAL MICA.



30 10 3 0 4 8 16

Tenth Normal HCl, cc. 2.5 Normal NaOH, cc.

FIG. 10.—THE ACTION OF HYDROCHLORIC ACID AND SODIUM HYDROXIDE UPON COMMERCIAL DRY-GROUND ORTHOCLASE.

on the surface of the mineral grains, and so protect the grains from the solvent action of water. Such compounds are lime, and the sulphates and chlorides of calcium and magnesium. If desirable, they may be removed later by such salts as sodium oxalate and sodium carbonate. The sulphates may be removed by barium carbonate, etc. Such removal, however, is not likely to be economical.

When it is desired to leach a slime in which the proportion of colloid matter is so high as to render it difficultly pervious to water, the grains may be separated from each other by further dilution and beating to a thin slip, or the slime may be preheated. Preheating a slime or clay is similar in nature to the coking of glue or starch. Water, etc., are driven off from the colloid *gel*, and the dehydrated *gel* "sets" to a hard, porous mass, which is easily leached. According to the work of A. V. Bleininger, who is in charge of the Clay Products Section of the Technologic Branch, from 250° to 300° C. is about the upper limit of temperature required for a favorable action. In Bleininger's investigation he successfully applied preheating to improve sticky clays that would otherwise crack excessively when wares made of them were dried.

It seems then that this development of the fundamental reactions of mineral colloid chemistry should benefit not only the clay-workers, for whom it was undertaken, but also metallurgists, treating materials that have so many points of similarity. Any specific application will require careful consideration and preliminary tests by an able chemist familiar with the principles of colloid chemistry. These principles have already been outlined.[8] A fuller treatment of this subject, mainly from the clay-working standpoint, will be completed this year under the title, The Technical Control of Colloid Matter of Clays, to be published probably by the U. S. Bureau of Standards.

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[8] *Bulletin No.* 388, *U. S. Geological Survey* (1909).

Development of Hindered-Settling Apparatus.*

BY ROBERT H. RICHARDS, BOSTON, MASS.

(Pittsburg Meeting, March, 1910.)

THIS is in part a review paper, indicating the various steps that have been taken in developing hindered-settling apparatus, some of the standard data that have been obtained, and some of the conclusions one is led to as to the effect of it in improving concentrating-methods and machinery, and in part it brings in some unpublished work.

CAUSES OF LOSS IN ORE-CONCENTRATION.

In making a study of the separation of valuable minerals from the waste gangue, particularly by water concentrators, it is necessary to define, first of all, what are the causes of loss that are likely to interfere with the concentration. Having made this definition, we can then study to see in what way the machines make their losses, and plan remedies for them in an intelligent way. Losses may be stated to occur as included grains, as flat or long grains, as fine free mineral, and as absolute slime. We will take these up one by one, as follows:

Included Grains.—Fig. 1 shows by a set of diagrams how included grains may occur: first, where the heavy mineral grains attached to the quartz are of large size; secondly, where they are of small size; and thirdly, where very fine particles of heavy mineral are included in the quartz. We have to decide in regard to included grains whether they are rich enough to go direct to the furnace, or poor enough to throw away, or whether they lie between these two points and require recrushing as the only means of severing the values from the waste. This last case is the only one requiring special treatment at our hands. We will therefore consider that next.

* By mutual consent, this paper has been presented also before the Western Society of Chemists and Metallurgists, and the Canadian Mining Institute, Montreal.

It will be noted that the valuable mineral is in coarse, medium, or fine grains, and that in order to sever these from the quartz, so that the concentrator can save them, we have to decide how fine the recrushing shall be. The exact size to which recrushing should be carried has to be determined in each case, and may be stated to be that size which liberates the maximum of the valuable mineral while making the minimum of slime. Where valuable mineral severs easily from the gangue, if it is brought down to twice the size of the grains of the valuable mineral, then they may be nearly all severed. If, however, the grains of valuable mineral do not sever easily from the gangue, then it may be necessary to break down to half their size in order to liberate a sufficient number of them completely from the gangue, so that the concentrating-machines

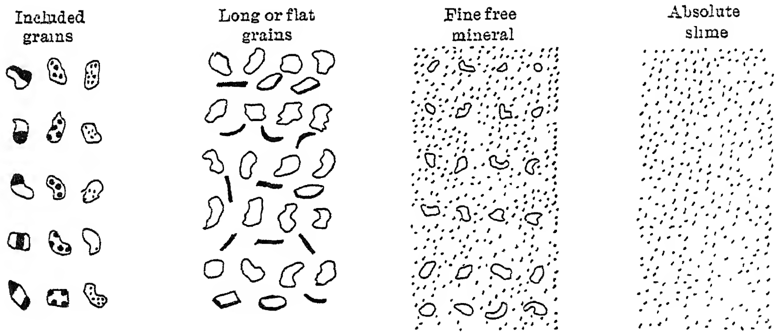


FIG. 1.—FOUR WAYS OF LOSING VALUES.

can satisfactorily do their work. It follows from this that where the valuable mineral is in large grains coarse crushing is suitable; where it is in medium-sized grains, crushing to medium sizes would be required; and where the grains are very fine, extremely fine crushing would be required.

Flat Grains and Long Grains.—Of these two, flat grains seem to be much harder to save than the long grains, as the flat grains have such a large surface in comparison with their weight that they settle slowly in water and therefore are not placed where their gravity would seem to indicate that they should be. There seems no way of saving flat grains other than to recrush them and treat them just as if they were included grains. The size of recrushing here will have to be determined by the same consideration as was shown under In-

cluded Grains. Graphite, being lighter than quartz, is an exception to the above rule; its flat form helps save it.

Fine Free Mineral.—Products containing fine free mineral, see Fig. 1, are made by jigs and Wilfley tables which have a mixed feed or a feed consisting of the finest products of classifiers; the tailings of vanners treating the ordinary stamp-mill pulp also contain fine free mineral. In my opinion, these three products, namely, jig-tailings, Wilfley-tailings, and vanner-tailings, furnish the ripest field for improvement in milling. The object of this paper is to set forth the methods of saving this fine free mineral.

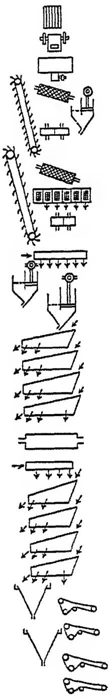
Absolute Slime.—This is the title that has been given to us by David Cole, of the Cananea Copper Co., and by it he means the grains of fine free mineral which are so fine that they cannot be settled in a reasonable commercial time. These grains cannot be handled, therefore, by settling. They can be handled to only an imperfect degree by the round table or by any of the film sizing-machines. Perhaps the canvas table is the most successful, and we may say that it is possible that some of the flotation-processes may succeed in saving this class of grain. The matter, in my belief, is still unsettled as to whether they can be saved by any means or not. The Anaconda method of allowing them to go out in a settling-pond, where the bulk of them settle in sufficient time to the bottom of the pond, is one method of handling this product, but owing to the low percentage of values and the large quantity of gangue, it cannot be said to be a satisfactory solution of the problem.

Typical Flow-Sheet.

Let us take up a general flow-sheet containing the chief machines used in concentrating, placing them in logical order, and briefly discuss the position of each, see Fig. 2.

The grizzly and the breaker, let us assume, have brought the material to 25 mm., or 1 in., in size, and have fed it to the mill-bin. The feeder next sends it to a trommel with 8-mm. holes. The oversize of this trommel is fed to the bull-jig, which therefore treats 25-mm. to 8-mm. material, yielding concentrates and middlings, or included grains requiring recrushing. The undersize of the trommel is fed to the next trommel, with holes 2.5 mm. in size. The oversize of this goes to Hancock jig,

which delivers concentrates, middlings for recrushing or to go back over the jig, and tailings either for waste or for recrushing, according to the richness and value of the ore. The undersize of this 2.5-mm. screen goes to a classifier and the coarsest two sizes go to jigs, which yield concentrates, and may yield tailings as waste and middlings for recrushing, or may yield only middlings for recrushing. The remaining four products from the classifier go to Wilfley tables and yield con-



Grizzly 25 mm	
Breaker. 25 mm.	
Bin. 25 to 0 mm.	
Feeder.	
Trommel. 8 mm.	
No. 1 Bull-jig. 25 to 8 mm. C. M. T	
Rolls. 8 mm.	
Elevator to 8-mm. trommel.	
Trommel. 2.5 mm.	
No. 2 Hancock jig. 8 to 2.5 mm. C. M T.	
Rolls. 2.5 mm.	
Elevator to 2.5-mm. trommel.	
Classifier. 2.5 to 0 mm. Making six products.	
No. 3 Jig on coarsest product.	C. M. T.
No. 4 Jig on next product.	C. M. T.
No. 1 Wilfley table with raff-wheel on next product.	C. M. T.
No. 2 Wilfley table on next product.	C. M. T.
No. 3 Wilfley table on next product.	C. M. T.
No. 4 Wilfley table on finest product.	C. M. T. Sl.
Tube-mill, Huntington or Chile mill to 1.25, 1, or 0.5 mm.	
Classifier. 1.25 (?) to 0 mm. Making four products.	
No. 5 Wilfley table {with raff-wheel and little classifier} on coarsest product.	C. M. T.
No. 6 Wilfley table on next product.	C. M. T.
No. 7 Wilfley table on next product.	C. M. T.
No. 8 Wilfley table on finest product.	C. M. T. Sl.
No. 1 Cone de-waterer on No. 4 Wilfley slime.	
No. 1 Vanner on No. 4 Wilfley tailings.	C. T.
No. 2 Vanner on No. 4 Wilfley slime.	C. T.
No. 2 Cone de-waterer on No 8 Wilfley slime.	
No. 3 Vanner on No. 8 Wilfley tailings.	C. T.
No. 4 Vanner on No. 8 Wilfley slime.	C. T.

FIG. 2.—TWO-MINERAL FLOW-SHEET.

centrates, middlings, and tailings. Where Wilfley tables are fed with the products of a good classifier, and are engaged in making a two-mineral separation, the use of a raff-wheel for returning the middlings to the table is not only a safe procedure, but wise, as the middlings form a very small product and one that can be perfectly separated by the table. Any middlings made by the two jigs or by the coarsest Wilfley tables which contain a large share of included grains, and therefore need

recrushing, may be broken by Chile mill, Huntington mill, or tube-mill down to 1.25 mm., or even to 0.25 mm., in diameter, according to the size to which the included grains need to be broken. The pulp from this regrinding must go to another classifier, followed by Wilfley tables, and the final winding up at the fine end can be made by sending the tailings of the finest Wilfley tables of both classifiers to a Callow screen, the oversize of which is clean gangue and the undersize makes a very perfect feed for the smooth-belt vanner; the slime from the finest Wilfley tables of both classifiers can be carried to a Callow cone and be finished up on vanners or other suitable machines. The tailings of these last vanners treating the fine slime will be the worst, and, in fact, the only unsatisfactory, product that will go out from such a mill. I will next take up the investigations that have been made and will refer to four investigations of the separation of sands by the free-settling method.

Free-Settling Tests.

1. In Close Sizing Before Jigging,[1] using a tube 9 ft. long and about 2 in. in diameter, dropping about 50 grains together, free-settling velocities were carefully determined. Tables I. and II. give, for grains ranging from 1.84 mm. to 0.106 mm. in diameter, the ratios of diameters for grains of quartz and galena when they settle at the same rate.

Table I.—*Diameters Corresponding to Velocities of Fall. The Fastest Grains only are Used for this.*

		Velocity in mm. Per Second.								
		25.4	50.8	76.2	101.6	127 0	152.4	177 8	203 2	228.6
		Diameter in Millimeters.								
	Sp. Gr.									
Anthracite..............	1.473	0.28	0.686	1.244	2.030					
Quartz	2.64	0.1399	0.2415	0.2795	0.432	0.762	1.041	1.346	1.650	1.905
Epidote	3.38	0.089	0.178	0.2665	0.381	0.508	0.648	0.864	1.040	1.295
Blende....................	4.046	....	0.165	0.2665	0.368	0.470	0.635	0.800	0.978	1.220
Pyrrhotite.............	4.508	....	0.140	0.216	0.292	0.381	0.470	0.597	0.762	0.914
Magnetite........	4.987									
Chalcocite.......	5.334	....	0.127	0.190	0.268	0.368	0.457	0.559	0.665	0.876
Arsenopyrite	5.627	.. ..	0.127	0.178	0.228	0.315	0.406	0.495	0.571	0.648
Cassiterite..	6.261	....	0.114	0.156	0.216	0.280	0.355	0.444	0.533	0.610
Antimony..............	6.706	....	0.089	0.140	0.216	0.280	0.355	0.444	0.546	0.635
Wolframite..	6.937	. . ..	0 089	0.152	0.208	0.267	0.343	0.419	0.495	0.585
Galena	7.586	....	0.089	0.152	0.190	0.254	0.305	0.368	0.432	0.507
Copper..................	8.479	....	0.089	0.140	0.183	0.254	0.325	0.376	0.432	0.507

[1] *Trans.*, xxiv., 409 to 486 (1894).

TABLE II.—*Free-Settling Factors, or Multipliers for Obtaining the Diameters of Quartz which will be Equal-Settling with the Mineral Specified when Settling Freely in Ample Water. Using Fastest Grains only.*

	Velocity in mm. Per Second.								
	25.4	50.8	76.2	101.6	127.0	152.4	177.8	203.2	228.6
	Multipliers.								
Anthracite	0.500	0.352	0.225	0.213					
Epidote	1.57	1.35	1.05	1.13	1.50	1.61	1.56	1.56	1.47
Sphalerite		1.46	1.05	1.17	1.62	1.64	1.68	1.66	1.56
Pyrrhotite		1.73	1.29	1.48	2.00	2.22	2.26	2.13	2.08
Chalcocite		1.90	1.47	1.62	2.07	2.28	2.41	2.44	2.17
Arsenopyrite		1.90	1.57	1.89	2.42	2.56	2.72	2.84	2.94
Cassiterite		2.11	1.79	2.00	2.73	2.93	3.03	3.05	3.12
Antimony		2.71	2.00	2.00	2.73	2.93	3.03	2.98	3.00
Wolframite		2.71	1.83	2.07	2.86	3.04	3.21	3.28	3.26
Galena		2.71	1.83	2.26	3.00	3.42	3.65	3.76	3.75
Copper		2.71	2.00	2.36	3.00	3.20	3.58	3.76	3.75

Example.—If a compact particle of galena, falling freely in water, settles 177.8 mm. per second, the particle of quartz of the same shape that will settle at the same rate will be approximately 3.65 times the diameter of the galena.

2. In the paper, Velocity of Galena and Quartz Falling in Water,[2] very complete figures of free-settling velocities of quartz and galena are given, ranging from 11.93 mm. in diameter to 0.00152 mm. in diameter, and these figures are the averages of 100 observations each.

In these tests the larger grains were dropped one at a time 2,800 mm. in a tube 8 in. in diameter, the intermediate grains were dropped 1,000 mm. in a tube 2.87 in. in diameter, and the smaller grains were settled 500 mm. in a decanting-tube (Bardwell's tube) 500 mm. high and 1.58 in. in diameter. While the ratios are somewhat different at the large end and also at the small end of the table, we find that the diameters of quartz grains from 2 to 0.5 mm. are about 3.0 to 3.5 times the diameters of the galena grains that settle at the same rate under free-settling conditions.

3. In the paper, Sorting Before Sizing,[3] mixed quartz and galena grains were fed to a glass *spitzlutte* with sorting-tube 24 in. high and $\frac{7}{8}$ in. in diameter. The heavy grains of quartz and galena that were drawn off below were caught in the little bulb

[2] *Trans.*, xxxviii., 210 to 235 (1908).

[3] *Trans.*, xxvii., 76 to 106 (1897).

placed there to receive them; the lighter grains which could not settle rose and went into the overflow. By using a series of currents starting with the strongest, rising 198.78 mm. a second, and ending with the weakest, 1.26 mm. a second, the overflow of a stronger current each time was fed to the *spitzlutte* with the next weaker current. The bulb-products so obtained from the series of rising currents form a very perfectly classified series of products, and present a very good opportunity for measuring with the microscope the average diameters of galena grains and of quartz grains found in each product (see Table III.). From these measured diameters we can again obtain the ratios of diameters under free-settling conditions. The results of the work are givén in Table III., and we see that the ratios of diameters for quartz grains from 1.97 to 0.589 mm. in diameter range from 3.70 to 2.92, or about from 3.5 to 3, as previously found.

TABLE III.—*Diameters of Quartz and Galena Particles which are Equal Settling in the Specified Upward Currents when Treated Under Free-Settling Conditions, together with the Diameter-Ratios.*

Diameter of Particles.		Particles Fall in Currents of.	Particles Rise in Currents of.	Ratio. Diameter of Quartz Divided by Diameter of Galena Averaged by a Curve.
Quartz.	Galena.			
mm.	mm.	mm. Per Sec.	mm. Per Sec.	
0.0301	0.0194	0.00	1.26	1.54
0.0335	0.0198	1.26	2.51	1.68
0.0568	0.0292	2.51	5.05	1.82
0.0772	0.0412	5.05	7.42	1.96
0.0982	0.0488	7.42	10.01	2.09
0.1423	0.0613	10.01	14.68	2.23
0.1875	0.0721	14.68	19.80	2.35
0.2254	0.1032	19.80	30.12	2.48
0.3416	0.1305	30.12	40.37	2.61
0.3880	0.1404	40.37	50.08	2.72
0.5241	0.1708	50.08	60.09	2.82
0.5892	0.1997	60.09	70.34	2.92
0.6590	0.2381	70.34	80.28	3.03
0.8604	0.2750	80.28	90.21	3.12
1.0234	0.3428	90.21	99.54	3.21
1.4224	0.3504	99.54	110.09	3.29
1.3216	0.3648	110.09	120.03	3.36
1.1424	0.3776	120.03	130.43	3.42
1.4256	0.4208	130.43	140.37	3.49
1.6032	0.4560	140.37	150.31	3.54
1.6848	0.4592	150.31	160.09	3.59
1.7488	0.4624	160.09	169.95	3.63
1.8032	0.5248	169.95	180.51	3.66
1.9746	0.5776	180.51	198.78	3.70

4. In the paper Free-Settling and Hindered-Settling Compared,[4] mixed quartz and galena grains were fed to an iron vortex-*spitzlutte*, Fig. 3, working under free-settling conditions. In each run of the series a weaker rising current was used than in the run just before it, and in each run the *spitzlutte* was fed with the overflow-product of the run just before it. In this way each run gave a heavy settling-product. These settling-products form a series ranging from large grains in the first to small grains in the last, and by sizing these products they can be laid out upon a board and photographed, see Fig. 4, which shows the little heaps of quartz and galena grains as obtained by free-settling. This photograph is of especial interest from two points of view:

First, we may, for any and all products, for example the product which settled against a rising current of 73.7 mm. a second, compute the average diameters of all the quartz grains, and the average diameters of all the galena grains, and if we divide the former by the latter we shall get the ratio of diameters 3.71 of the quartz and galena grains which settle together under free-settling conditions. If we calculate them for all the different velocities we get the results in Table IV.

TABLE IV.—*Free-Settling Diameter-Ratios of Quartz and Galena.*

Beginning heaps:						
Velocities, . .	200					
Ratios, . . .	3.208					
Middle heaps:						
Velocities, . .	144	103	73.7	52.8	37.9	27.2
Ratios, . . .	3.941	4.135	3.710	2.794	3.172	3.256
End heaps:				Rises in		
Velocities, . .	14.5	14.0	10.0	10.0		
Ratios, . . .	1.802	3.216	2.844	1.334		

The beginning heaps are too near the galena to be fair; the end heaps are too near the slime to be fair; the middle heaps represent the best we have for ratios. Here again we see the ratios run from 3 to 3.5, more or less, as the ratios for quartz and galena grains settling under free-settling conditions. The average of the ratios of middle heaps is 3.501.

The second matter that is of special interest is the relation of

[4] Not yet printed.

the two ranges of hills. Note that the range of quartz hills is very close to the range of galena hills with only a very slight valley between the two. This is the characteristic result of separating grains by a free-settling classifier. We have now reviewed four different methods of testing by free-settling.

1, by letting clusters of grains settle together,

2, by letting individual grains settle one at a time,

3, by *spitzlutte* and microscope to get the ratio,

4, by *spitzlutte* and photograph to get the ratio,

and we find that they all give very nearly the same ratios of diameter for galena and quartz that are equal settling, namely, about 3 to 3.5 for the range from 0.5-mm. quartz to 2-mm. quartz.

Free-Settling Mill-Classifiers.

To get the free-settling method of classifying into a practical mill form, there are four things to be provided for:

1. We must feed the pocket or bring in the mixed sands to be treated.
2. We must subject them to a rough sorting in the pocket, lifting out the finished light product or overflow, using the feed-water with the rising water from the sorting-column to do this work, and settling the heavy unfinished product to the sorting-column.
3. We must subject the above-named heavy product to a finishing-sorting in the sorting-column, allowing the finished heavy product to settle down and out, while the entangled light grains are separated and lifted up to the pocket again. This sorting-column must have the same velocity of rising current in all parts of a horizontal section across it.
4. Finally, we have a pressure-box into which the wash-water is admitted and where it divides itself into two parts, that which rises and does the work in the sorting-column and that which goes down and out the spigot with the discharged sand. This pressure-box is a species of ticket-office entrance-gate through which the heavy grains are allowed to pass only after they are up to the standard and have obtained their ticket in the sorting-column above.

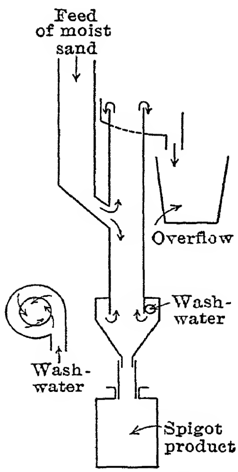


FIG. 3.—SPITZLUTTE WITH VORTEX.

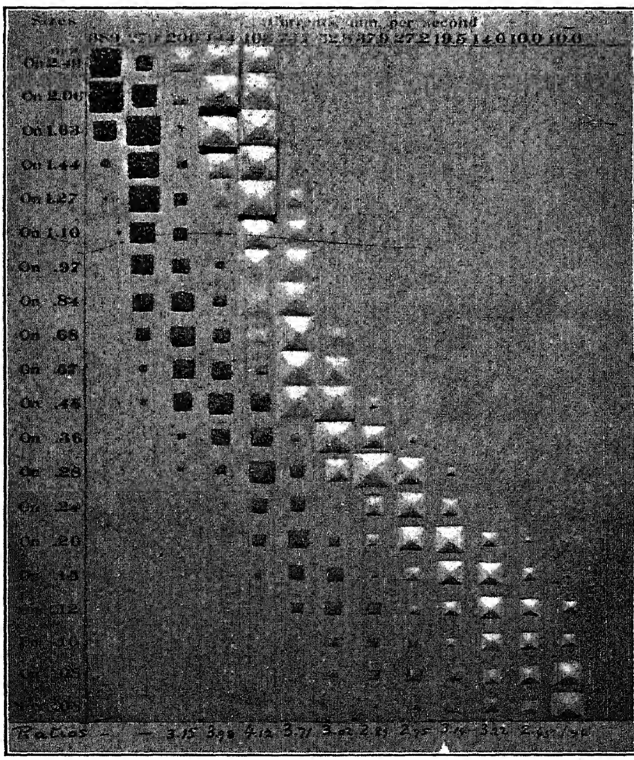


FIG. 4.—FREE-SETTLING TEST.

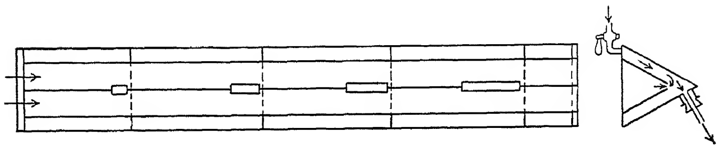


Fig. 5.—Hog-Trough Classifier.

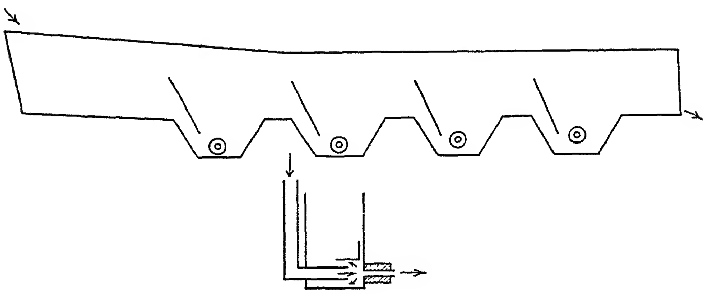


Fig. 6.—Calumet Classifier.

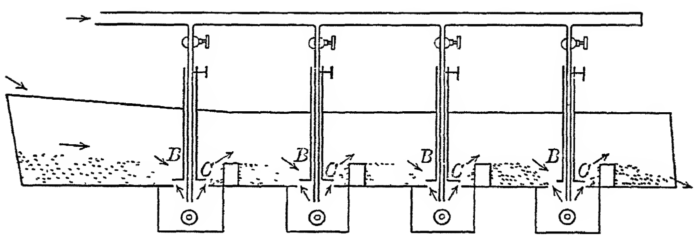


Fig. 7.—Evans Classifier.

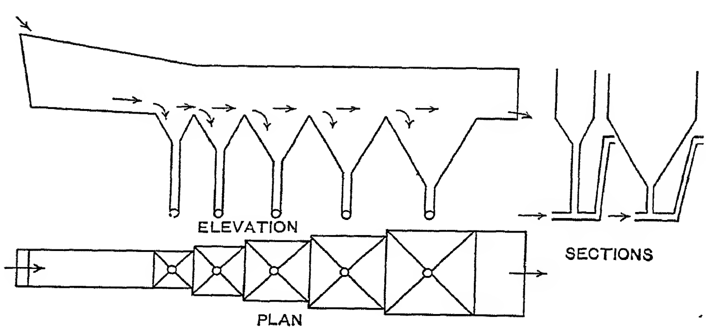


Fig. 8.—Wengler Classifier.

The sorting-column is the most difficult part of all the design, as it must not allow local down-currents which would carry slime down into the spigot-product, but it must have a steady, even rising current to carry up and out perfectly all the lighter grains that do not belong in the spigot-product. Classifiers are good or bad according as they do this one thing well or ill.

It may be further remarked that with all the pocket-classifiers, the water in the pocket must have the same rising-velocity as that in the sorting-column in order that they may both lift the same size of grain. For all these matters of computation of classifiers see my book.[5]

The Lake Superior Hog-Trough Classifier.—The advantage of this classifier, see Fig. 5, is its simplicity. Nothing is likely to happen to it that will put it out of order. The disadvantage of it is that it does not keep slime out of the spigot-products or prevent waste in fine free mineral in the jig- and Wilfley-table tailings. The fault is due mainly to two causes: (1) The sorting-column is only the thickness of a board in height, and, therefore, permits local down-currents to carry down slime. (2) The rising current is not positive in its action. When the water is increased by the cock the full effect does not take place until the water has risen in the annular space between the inner and the outer V to its full height, due to the increase. With this uncertainty, it is impossible to get good adjustments. Other harmful complications also arise.

The Calumet Classifier.—This has the "hydraulic" water flowing directly at and in line with the plug-spigot, see Fig. 6, and the effect of adjustment of water is instantly felt. The slime is kept out of the spigots by the fact that the sand has to pass through an annular conical zone of clear water in order to get out of the spigot. The classifier is simple and easily kept in order and loses little mill-head. This was the first positive classifier.

The Evans Classifier.—This has the advantage of simplicity, see Fig. 7, and of being easily kept in order. It has, however, one rather serious disadvantage in the fact that it requires greater force in the rising current on the up-stream side, *B*, of the regulating-disk, *C*, than on the down-stream side; this

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[5] *Ore Dressing*, vol. iii., p. 1375 (1909).

necessitates using more water on the down-stream side than is necessary for cleaning the sand, and makes the classifier a large water-user.

The Wengler Classifier.—This consists of hopper-shaped pockets, see Fig. 8, with feed and overflow of all on the same level. It has small vertical pipe sorting-columns at the points of the hoppers. This was a very early design, made before the classifier was understood. It was probably never put into use but once and then immediately thrown out. It has one fatal error: the hopper-shaped pockets with small sorting-columns below must necessarily fill up with hard sand-banks, for the grains of sand that are just too small to settle in the rapid current of the small sorting-column are much too large to be raised in the weak rising current in the large area at the top of the hopper-pocket. I have put this in simply as a warning to the would-be designer.

The Linkenbach Classifier.—This is one of a number of designs, see Fig. 9, given by Linkenbach in his book,[6] and shows a simple way of overcoming the trouble of the sand-banks referred to under Wengler. The method is to have the overflow of any one pocket above the level of the feed of the next. This gives a plunging-stream feed to all the pockets and breaks up the sand-banks, carrying forward the sand to the next pocket. This method is not systematic and may carry forward into the later pockets larger grains than belong there.

The Pocket Vortex-Classifier.—By having the pockets shallow and graded in size, Fig. 10, so that an earlier smaller pocket settling large grains will have a stronger rising current, while a later larger pocket settling smaller grains will have a weaker rising current, the roughing-work of the classifier of lifting out the most of the lighter grains is easily and perfectly done. The sorting-column below is put on to complete the work of cleaning the spigot-product. The hurtful down-currents carrying slime into the spigot are prevented by admitting the "hydraulic" water in a vortex fitting. This gives a helical rising-current, that is, a current flowing up in the direction of the thread of a screw. This classifier is simple and complete in its action.

The Tank Vortex-Classifier.—This classifier, Fig. 11, adds to

[6] *Aufbereitung der Erze* (1887).

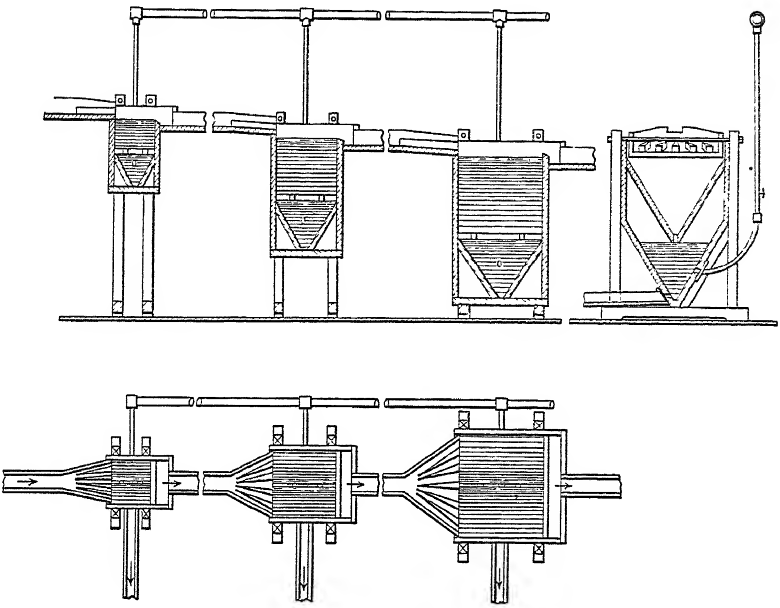

FIG. 9.—LINKENBACH CLASSIFIER.

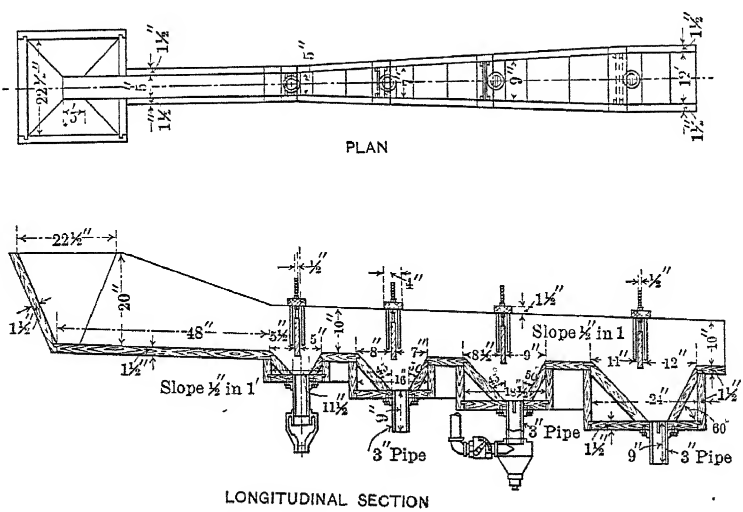

FIG. 10.—POCKET VORTEX-CLASSIFIER.

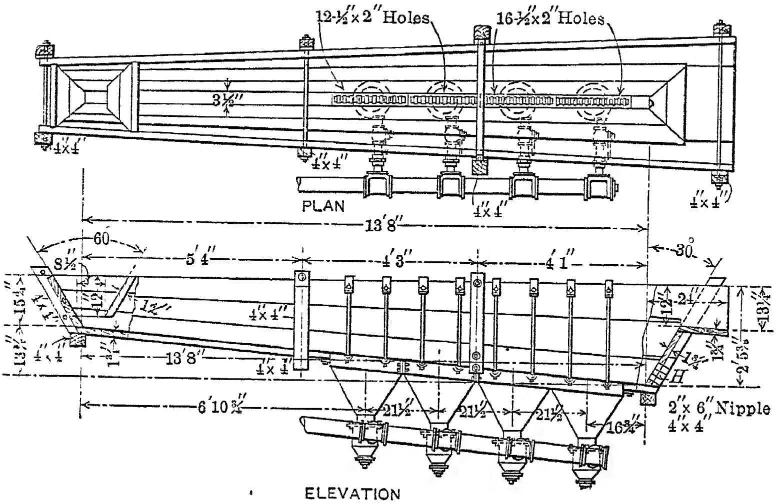

FIG. 11.—TANK VORTEX-CLASSIFIER.

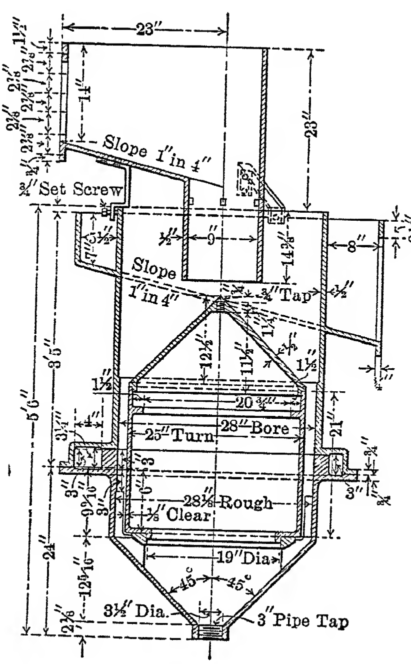

FIG. 12.—ANNULAR VORTEX COARSE CLASSIFIER.

the advantages of the pocket vortex-classifier the fact that all the pockets are merged into one tank, and no careful grading of pockets has to be allowed for. In case the design provides for a larger amount of feed-water than is used, then the resulting tendency to form a bank is overcome by a safety-spigot, *H*, in the end of the tank. This safety-spigot product ranks as overflow and may be treated with the latter or as a separate product, as preferred. It will have slime mixed with its fine sand. This classifier is simple and complete in its action.

The Annular Vortex Coarse Classifier.—This classifier, Fig. 12, is generally used with one pocket only, and has found employment in two directions: (1) for cleaning the fine overflow of the log-washer when used on brown iron-ore, where it saves 25 tons a day of good iron-ore for the furnace; (2) for separating coarse sand for feeding the Hancock jig from the fine sand for the Wilfley table. It makes use of the helical current of the vortex-classifier for very much larger quantities than can be used in a simple pipe sorting-column, and does so without the danger of hurtful local down-currents to carry slime into the spigot. In fact, it can be designed for almost any capacity.

The Annular Vortex Fine Classifier.—This classifier, Fig. 13, has an enlarged cone above the annular sorting-column to enable the classifier to deliver a fine overflow-product of any degree of fineness by the large diameter and consequent small velocity of the overflow. At the same time it delivers a spigot-product free from slime.

The Callow Slime-Cone and Water-Cone.—These cones, Fig. 14, are of two kinds: (1) the slime-cone, which draws sand and some slime from the spigot and sends slime into the overflow; and (2) the water-cone, which draws sand and slime from the spigot and clear water, or nearly clear water, from the overflow. They have no rising "hydraulic" water. They are really settling-tanks. The reason for mentioning them here is to bring out the advantage they possess over the rectangular settling-tank. An ordinary settling V-tank 6 ft. wide has an overflow 6 ft. wide; an 8-ft. water-cone has an overflow 24 ft. wide. If the two tanks have the same quantity (gallons a minute) to treat, the V-tank will have four times the amount of overflow per foot that the water-cone has;

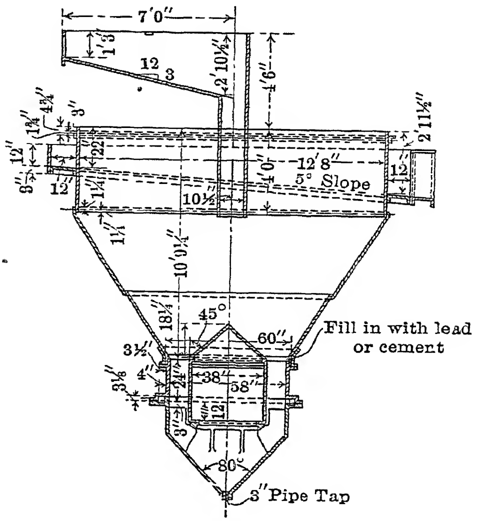

FIG. 13.—ANNULAR VORTEX FINE CLASSIFIER.

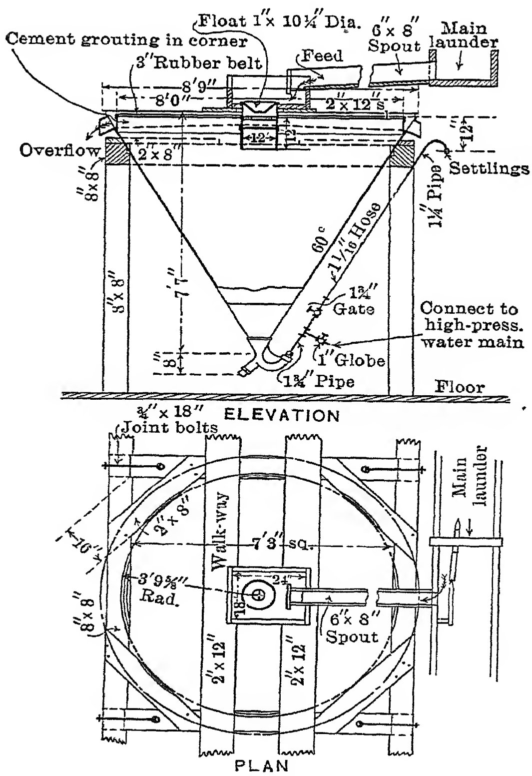

FIG. 14.—CALLOW SETTLING-CONE.

the water approaching the overflow will have four times the velocity, and will therefore carry much larger grains into the overflow.

THE WORK OF A FREE-SETTLING CLASSIFIER.

Fig. 15 is introduced to give more clearly the exact idea of what happens in a free-settling classifier. To the left of this figure will be seen a diagram representing a tall glass tube of water in which grains of sand can be allowed

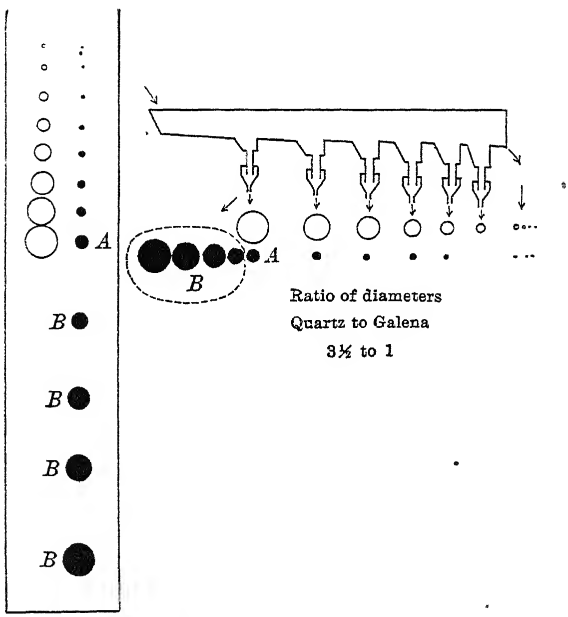

FIG. 15.—THE WORK OF A FREE-SETTLING CLASSIFIER.

to settle, and in which they will settle with velocities according to their size and gravity. The open circles at the left represent quartz with a gravity 2.64, and they show that the largest grain goes the fastest and the smaller grains diminish in velocity as their diameters diminish. Similarly, the black circles represent galena with a gravity 7.5. Here the grains also settle according to their size, the larger going faster, but the galena grains settle much faster than quartz of the same diameter. We at once notice that the grains are paired off, that there is in each pair a smaller grain of galena

opposite the larger grain of quartz which is equal settling with it, and the quartz grains are, as we have already seen, about 3.5 times larger in diameter than the galena grains. These grains having the same velocity are said to be equal-settling grains under free-settling conditions, and they naturally belong in the same spigot of the classifier. If now we suppose this complete group of grains to be laid in a horizontal position underneath the classifier, we shall see that each spigot receives its large grain of quartz and small grain of galena according to the equal-settling velocities that have been discussed in the early part of this paper, that in succession the second spigot has smaller grains than the first, the third has smaller grains than the second, and so on, diminishing until we reach the final overflow, which contains the finest grains of all; but the most striking thing is the remarkable difference shown by the first spigot, for besides the equal-settling grains of quartz and galena, *A*, we have all the added increment, *B*, of larger grains of galena, which find their way into this spigot in addition to the regular classified grains of quartz and galena, making the first spigot uniformly very much richer than any of the subsequent spigots. A supplementary condition occurs also in the overflow. Here the largest grain of quartz is much larger than the largest grain of galena, and this portion is the truly-classified product; but in addition to this the quartz ranges down to the finest dust, and the galena also making this product has an added increment of extra-fine grains over and above the truly-classified grains, making it differ from the truly-classified products of all the intermediate spigots.

We see that all of the products of the classifier, except the first spigot and the last overflow, are truly-classified products; that is to say, they contain larger grains of quartz and smaller grains of heavy mineral, but that the first has an added increment of a large quantity of coarse grains of the heavy mineral, which enrich it very largely, while the last overflow has an added increment of fine material over and above the truly-classified grains. These facts showing the difference between the coarsest and finest products render it necessary to give these two products special consideration when they are being treated by concentrating-machines.

Hindered-Settling Tests.

Hindered-Settling with Steady Current.—If in Fig. 16, after turning the water on moderately by the valve, *G*, filling the glass tube and hopper to the overflow, we charge mixed quartz and galena grains from 2 mm. to 0 in size, and as the rising current of water is too strong to allow them to go through the 0.25-in. tube constriction at the bottom, the grains will quickly arrange themselves according to their ability to settle under these conditions, fine grains at the top, coarse grains at the bottom, and it will be found that they are all in movement, teetering or oscillating up and down through short distances, like the grains of sand in a boiling spring. If one should take out a grain from any part of the tube and drop it in at the top, it would settle down until it came to approximately the same position as that from which it was taken. These grains have arranged themselves according to the laws of hindered-settling particles. If now we slacken the water and draw off a bulb full of sand, and we do it a second time with a new bulb, and a third time and so on, we get a series of products beautifully classified. We may now size these products on our series of sieves and lay them out on a board and photograph them, Fig. 17. After doing this, we can compute for any bulb the average diameter of the quartz grains, and the same for the galena grains, and divide the former by the latter and get the diameter-ratio for quartz and galena under hindered-settling conditions. Such a computation for bulbs 4, 5, and 6 gave the results in Table V.

TABLE V.—*Hindered-Settling Diameter-Ratios of Quartz and Galena. First Test.*

Number of bulb,	4	5	6	Average.
Ratio of quartz to galena,	6.325	5.656	5.544	5.84

This (First discovery) shows a remarkable advance from the diameter-ratio of free-settling. See also the wide valley between the quartz and galena heaps in Fig. 17. The same work has been done for the 13 minerals referred to under Free-Settling, and the hindered-settling diameter-ratios given in Table XII. were obtained. See the paper, Close Sizing Before Jigging.[7]

[7] *Trans.*, xxiv., 409 to 486 (1894).

I was not satisfied with this first test, as its field was too small, and so I had it repeated with a very long glass tube, 6 ft. long, Fig. 18, which yielded 45 bulbs, and from them, after sizing, the photograph, Fig. 19, was obtained. Diameter-ratios were computed from these bulbs and gave Table VI.

TABLE VI.—*Hindered-Settling Diameter-Ratios of Quartz and Galena. Second Test.*

Heavier products:

Bulbs,	12	13	14	15	16	17	18
Ratios,	4.6	5.19	5.68	5.72	6.03	6.11	6.07

Intermediate products:

Bulbs,	19	20	21	22	23	24	25	26	27	28
Ratios,	6.73	7.22	6.93	6.95	6.93	7.50	6.91	6.39	8.00	6.60
Bulbs,	29	30	31	32	33	34				
Ratios,	7.36	6.24	6.30	6.95	6.75	6.97				

Light products:

Bulbs,	35	36	37	38	39	40	41	42	43	44	45
Ratios,	5.88	6.12	5.66	4.53	4.75	4.48	4.44	4.50	4.15	1.50	1.77

Throwing out of the count the first few heavier products as being too near the galena to be fair, and also a few of the light products as being too near the slime to be fair, we get the above-mentioned ratios from the intermediate products, and their average is 6.92, showing a still greater advance over free-settling diameter-ratio than was shown in the first hindered-settling test. Observe also the wonderfully wide valley between the quartz and galena heaps in Fig. 19.

Hindered-Settling with Pulsating Current.—With a view to see whether anything was gained by the use of intermittent pulsions in the place of steady current under hindered-settling conditions, the following experiment was tried: Fig. 20 shows a hindered-settling tube which has been cut apart and a sieve has been placed across it as a horizontal jigging-sieve. Tests made with this tube indicated that the ratio of diameters was not advanced over that obtained by the steady current; and photograph, Fig. 21, from this work is identical with photograph, Fig. 17, without pulsations. The tests, however, indicated that the pulsating current could be made to act much more evenly than the steady current; that is to say, the avoidance of downward hurtful currents could be gained to a much greater

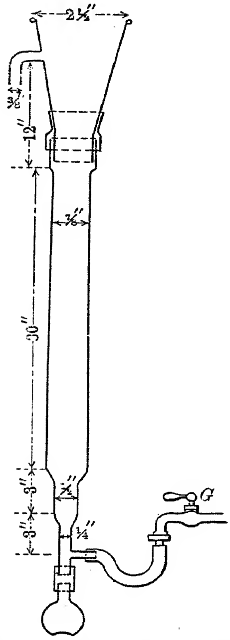

FIG. 16.—POINTED TUBE, OR HINDERED-SETTLING SPITZLUTTE.

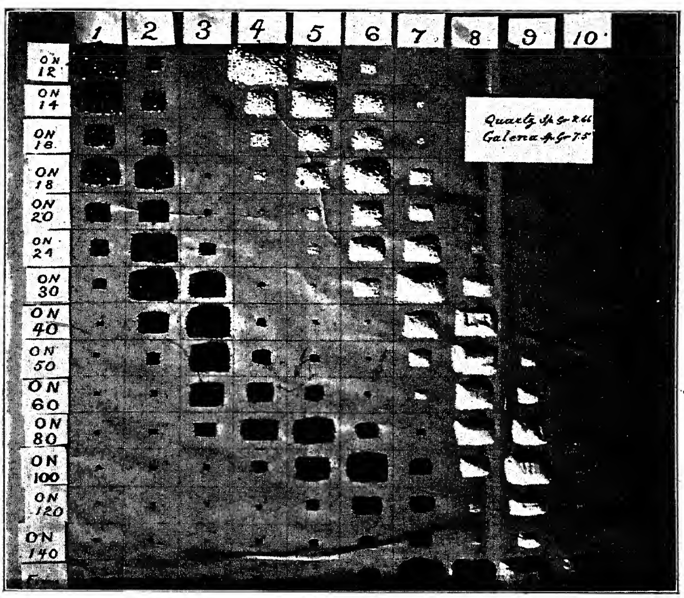

FIG. 17.—HINDERED-SETTLING TEST WITH POINTED TUBE.

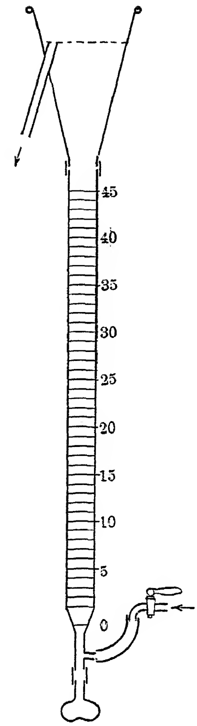

Fig. 18.—Lewis and Richmond Hindered-Settling Tube with Bulb Drawn 45 Times.

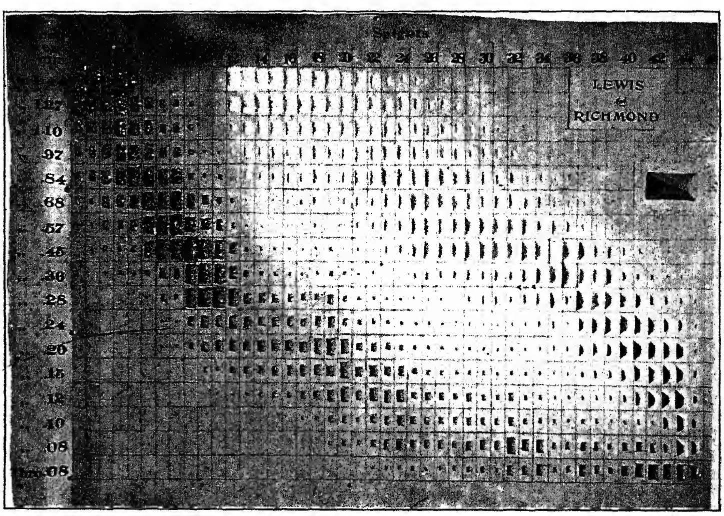

Fig. 19.—Hindered-Settling Test with Long Tube.

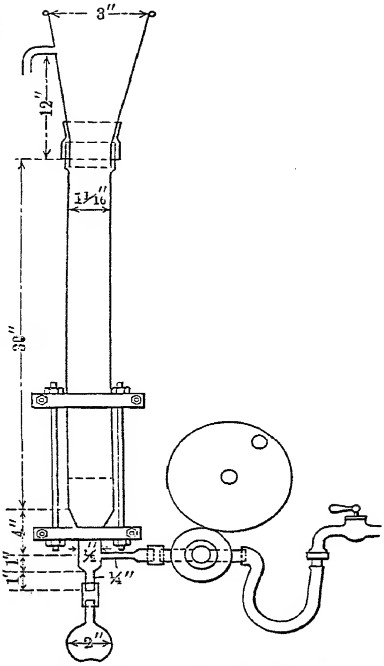

FIG. 20.—PULSATOR-JIG.

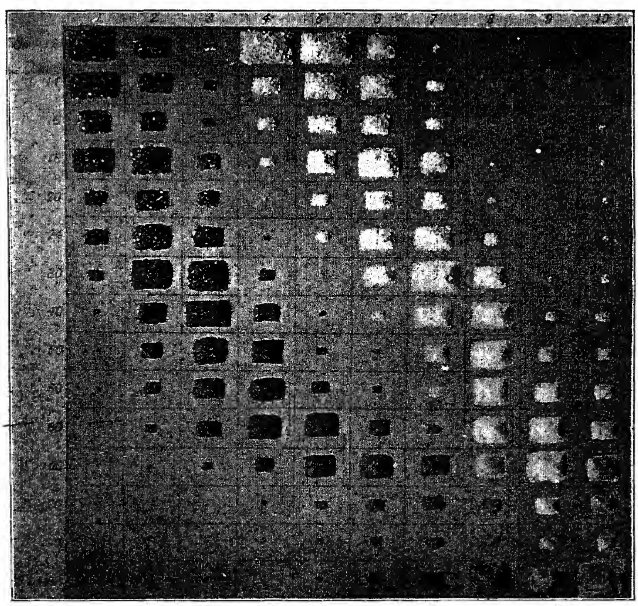

FIG. 21.—HINDERED-SETTLING TEST WITH PULSATOR-JIG.

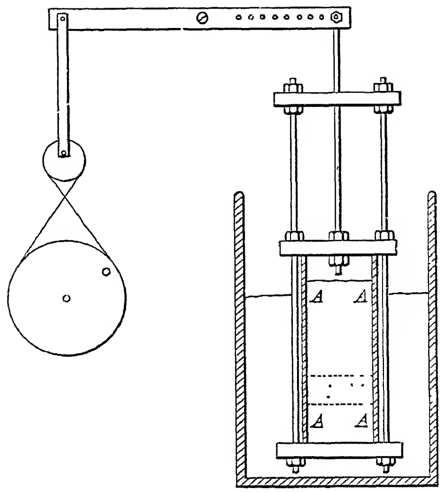

FIG. 22.—HAND-JIG.

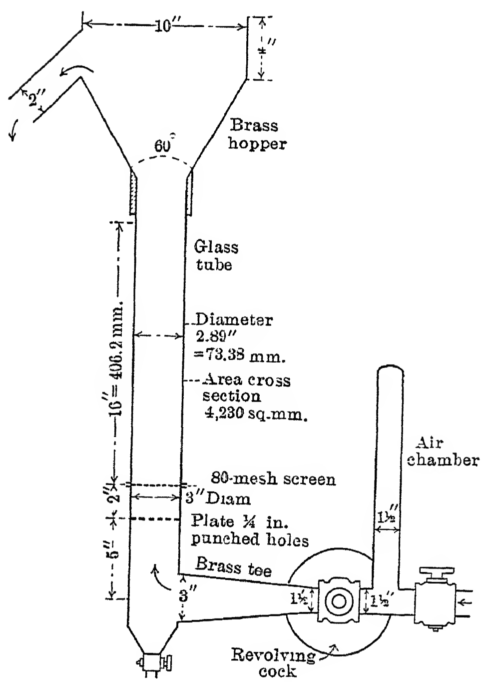

FIG. 23.—APPARATUS FOR STUDYING HINDERED-SETTLING CONDITIONS.

degree by the use of a pulsating current acting evenly all over the surface of the sieve than could be done with a steady current, in both cases acting under hindered-settling conditions.

Method of Getting Hindered-Settling.

It will be seen that the hindered-settling apparatus has a constriction whether by narrowing the tube or by the use of a sieve, up through which the rising current comes, and that the tube widens out to a larger diameter and area in the hindered-settling chamber above the constriction. This is the essential feature of the method of getting hindered-settling, namely, a hindered-settling chamber above and a constriction below it, with the whole current rising up through the two.

Cause.—The cause of the difference between hindered-settling and free-settling appears to be that with free-settling both the quartz and the galena are settling in water with a specific gravity of 1, and they obey the law laid down by Rittinger for particles settling in that liquid. With hindered-settling, however, the grains are settling in liquid much denser than water, with perhaps 1.5 specific gravity, due to the crowding mineral grains. This, when worked out mathematically, should give a diameter-ratio of 6.9 between quartz and galena, and this is exactly what has been obtained in Fig. 19.

Effect.—The effect of hindered-settling and the advanced diameter-ratio is twofold: first, the slime is kept out of the spigot-product more perfectly than by any other method known to me; second, the advanced ratio really means that the hindered-settling classifier has done more to help the concentrating-machines which follow to do their work than has the free-settling classifier. In proof of this, the reader is asked to compare the hindered-settling photographs, Figs. 17, 19, and 21, with the free-settling photograph, Fig. 4. These photographs all represent the work of a classifier as a preliminary machine followed by screens as the finishing machine. Notice how much farther the galena is removed from the quartz in the hindered-settling photographs than in the free-settling photograph. Notice also how much more galena is concentrated into the first spigot of the hindered-settling than is in the free-settling classifier.

Requirements.—As the teeter-chamber (hindered-settling

chamber) must have a constriction below it, it is necessary to know what should be the ratio of areas between the two. Clearly it cannot be so large a ratio as in Fig. 16, namely, $(\frac{1}{4})^2$ to $(\frac{7}{8})^2 = 12.25$, because when the current was strong enough to cause full teeter in the teeter-chamber it was too strong in the constriction to allow any grain to go down, and it cannot be a very small ratio, as then it would simply be a free-settling classifier. Clearly there must be some ratio that is better than all others. I have worked long at this point, and I think I now have end figures in sight. Up to date I can say that a ratio of areas of 1 to 4 will work, that is, it will give a teeter-chamber above and a discharge through the constriction at the same time.

The Jig.

From what has already been said it will be understood that the act of pulsion or the movement of rising current in jigging is identical in its action with the hindered-settling rising current. The plunger-jig, however, adds another very important feature, namely, the act of suction, or return-downward current. This is caused by the rising of the plunger when returning preparatory to the next downward or pulsion stroke.

In the paper, Close Sizing Before Jigging,[8] three series of tests were recorded which threw a great deal of light on this question.

1. A little glass hand-jig, Fig. 22, was used in such a way that the water at no time reached the top of the glass side, *A*, *A*. The lift-pump valve-action of the sand grains soon brought the water to its maximum height within the glass, and then it was evident that the amount of water that rose through the sand during pulsion was equal to the amount of water that fell during suction; that is, we had jigging with strong suction.

2. The above glass jig was filled up with water until the water stood above the glass sides, *A*, *A*, of the jig. When so adjusted a considerable portion of the water which rose during pulsion could overflow, and therefore the amount of water which came down during suction was very much reduced. It was then jigging with mild suction.

3. The glass pulsator-jig, Fig. 20, of this paper has pulsion only, and therefore represented jigging with no suction.

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[8] *Trans.*, xxiv., 409 to 486 (1894).

Six tests were made in each series with each of the above three sets of conditions, using the same size of quartz in all cases, but varying the size of blende from coarse to fine in each of the three sets.

The results are given in Table VII.

TABLE VII.—*Jig-Tests on Quartz and Blende.*

Diameter of the Grains.

	Inch.	Inch.	Inch.	Inch.	Inch.	Inch.
Diameter of quartz........	0.0683	0.0683	0.0683	0.0683	0.0683	0.0683
Diameter of blende.......	0.0683	0.0429	0.0262	0.0195	0.0095	0.0042
Series 1, with strong suction.						
Pulsions needed for separation	2,129	1,676	1,757	297	208	288
Per cent. blende brought down........................	96	95	95	95	99	99
Series 2, with mild suction.						
Pulsions needed for separation................	306	838	846	1,382	1,729	∞
Per cent. blende brought down......................	99	99	100	98	97	0
Series 3, with no suction.						
Pulsions needed for separation................	147	202	496	∞	∞	∞
Per cent. blende brought down.......	98	95	50	0	0	0

The results show that with a closely-sized product, 0.0683-in. quartz and 0.0683-in. blende, strong suction is very slow (2,129 pulsions needed), while no suction is extraordinarily rapid, 147 pulsions needed (Second discovery). On the other hand, with second spigot of a classifier where the diameter of the quartz is 3.5 times the diameter of the heavy mineral ($0.0195 \times 3.5 = 0.0683$), strong suction is very rapid, 297 pulsions needed, while no suction would never make a separation (∞ means infinity or never). Mild suction appears not to have been very satisfactory in either case.

Full Teeter.—When a water-current rises, Fig. 23, through a mass of sand where the grains are all of one size, if very slow it will not move the grains at all; if faster, the grains begin to move. By gradually increasing the current, we reach a speed

where all the grains are moving so freely that they are immediately stratified according to gravity. When this speed is reached, we call the condition "just full teeter," that is, the weight of the grains is balanced by the rising water-current, and as a result, they are teetering up and down like the grains of sand in a boiling spring.

The data in Tables VIII., IX., X., and XI. are the result of many trials, some of them as many as 20, before adopting the results.

TABLE VIII.—*Hindered-Settling Figures for Galena, Specific Gravity* 7.5. *Area of Cross-Section of Tube*, 4,230 *sq. mm.*

Size of Grain.			Still Height.	Just Full Teeter Height.	Just Full Teeter, Hindered-Settling Velocity. If no Sand.	Free-Settling Velocity.[a]
Maximum.	Mimimum.	Average.				
mm.	mm.	mm.	mm.	mm.	mm-sec	mm-sec.
7.66	5.58	6.67	108	165.	212.4	697.7
5.58	4.06	4.97	112	164.3	183.7	590.3
4.06	2.74	3.40	121	167.5	141.9	505.1
2.74	1.95	2.35	119	170.0	118.7	401.8
1.95	1.44	1.70	125	165.7	90.42	350.2
1.44	0 97	1.21	124	175.0	77.18	272.8
0.97	0.68	0.83	127	177.2	51.84	213.1
0.68	0.45	0.565	126.5	180.34	36.33	175.3
0.45	0.36	0.405	126.5	186.18	15.40	124.8
0.36	0.24	0.300	129.5	212.09	17.59	115.5
0.24	0.15	0.195	136.4	219.20	10.32	77.08
0.15	0.00	0.0705	151.3			28.44

[a] Very good figures. From the paper, Velocity of Galena and Quartz, *op. cit.*

TABLE IX.—*Hindered-Settling Figures for Blende, Specific Gravity* 4.00. *Area of Cross-Section of Tube*, 4,230 *sq. mm.*

Size of Grain.			Still Height.	Just Full Teeter Height.	Just Full Teeter, Hindered-Settling Velocity. If no Sand.	Free-Settling Velocity.[b]
Maximum.	Minimum.	Average.				
mm.	mm.	mm.	mm	mm.	mm-sec.	mm-sec.
5.58	4.06	4.97	115.0	165 7	125.9	....
4.06	2.74	3.40	123.0	182.0	91.33	......
2 74	1.95	2.35	128.0	187.0	70.65	....
1.95	1.44	1.70	131.0	191.0	58.75	263.0
1.44	0.97	1.21	131.0	198.0	43.38	227.0
0.97	0.68	0.83	134.0	208.0	35.15	181.0
0.68	0.45	0.565	138.0	216.0	24.59	143.0
0.45	0.36	0.405	140.0	238.0	16.82	109.0
0.36	0.24	0.30	143.0	256.0	10.96	86.0
0.24	0.15	0.195	147.0	275.0	6.70	59 0
0.15	0.00	0.075	157.0			

[b] From the paper, Close Sizing Before Jigging (*op. cit.*). Not as good figures as quartz and galena.

TABLE X.—*Hindered-Settling Figures for Quartz, Specific Gravity 2.64. Area of Cross-Section of Tube,* 4,230 *sq. mm.*

Size of Grain.			Still Height.	Just Full Teeter Height.	Just Full Teeter, Hindered-Settling Velocity. If no Sand.	Free-Settling Velocity.
Maximum	Minimum.	Average.				
mm.	mm.	mm.	mm.	mm.	mm-sec.	mm-sec.
7.66	5.58	6.77	128	200	115.9	295
5.58	4.06	4.97	129	211.2	112.6	249
4.06	2.74	3.40	128.27	171.0	68.72	207
2.74	1.95	2.35	130.3	171.0	50.17	166.5
1.95	1.44	1.70	130.05	189.2	41.41	134.0
1.44	0.97	1.21	129.7	190.6	32.48	106.0
0.97	0.68	0.83	133.35	202.4	20.79	80.5
0.68	0.45	0.565	139.7	296.5	12.89	59.5
0.45	0.36	0.405	140.97	....	....	41.5
0.36	0.24	0.300	136.4	249.6	6.48	33.7
0.24	0.15	0.195	148.6	277.6	3.31	23.0
0.15	0.00	0.075	139.8	....	....	....

TABLE XI.—*Hindered-Settling Figures for Anthracite Coal. Area of Cross-Section of Tube,* 4,230 *sq. mm.*

Size of Grain.			Specific Gravity [a]	Still Height.	Just Full Teeter Height.	Just Full Teeter, Hindered-Settling Velocity. If no Sand.	Free-Settling Velocity.
Maximum.	Minimum.	Average.					
mm.	mm.	mm.		mm.	mm.	mm-sec.	mm-sec.
11.01	7.66	9.34	1.452	122	205	104.6	
7.66	5.58	6.62	1.463	124	195	71.63	Have no figures on velocities.
5.58	4.06	4.97	1.444	128	195	53.26	
4.06	2.74	3.40	1.683	128	187	41.37	
2.74	1.95	2.35	1.695	130	193	32.55	

[a] The coal varied in purity, and therefore the specific gravity is given for each size.

Grains of quartz 3.4 mm. in diameter, Table X., require for just full teeter a rising current of 68.7 mm. a second (that is, a quantity of water that would give this velocity if no sand were present), while these same grains under free-settling conditions settle at the rate of 207 mm. a second. Galena grains 3.4 mm. in diameter, at just full teeter, see Table VIII., require a rising current of 141.9 mm. a second, while the same grains settle under free-settling conditions at the rate of 505 mm. a second. Let us see to what conclusion these interesting facts lead us. See Fig. 24. If we are concentrating 3.4-mm. quartz and galena grains in a pulsator-jig, we must give the galena 141.9 mm. a second velocity of rising current in order that they may

be at full teeter or liquid stratification. This 141.9 mm. a second is far above the just-full-teeter velocity of quartz, in fact it is nearly up to the free-settling velocity of quartz. We see, then, that when galena is in hindered-settling condition at just full teeter the quartz is so separated that the grains are far apart. We have in this phenomenon the answer to the question, why is it that the pulsator-jig can treat ore so much more rapidly than the plunger-jig? for in the pulsator-jig a grain of galena meets no resistance to settling through the quartz layer

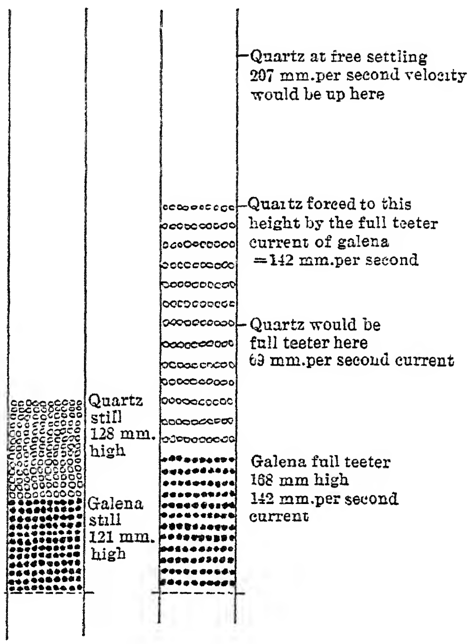


Fig. 24.—Behavior in Pulsator-Jig of Galena and Quartz 4.06 to 2.74 mm. in Diameter.

lying above the galena (Third discovery), while on the other hand, if we drop a grain of galena on the bed of a plunger-jig and watch it, it takes 5 or 10 pulsions before it settles into the quartz far enough to get out of sight. This clearly explains the remarkably high capacity of the pulsator-jig.

Economy of water should be largely realized by the use of hindered-settling. Since it requires only one-quarter, more or less, of the velocity required by free-settling to do the work, it should require only one-quarter the quantity of water. As

the designs are being improved this economy is more and more being realized. For example, in the direct pulsator-classifier, Fig. 29, the quantity required is extraordinarily small, and in the pulsator-jig the water consumed is less than in the plunger-jig. The No. 1, No. 2, and No. 3 hindered-settling classifiers, Figs. 25, 26, and 27, have not yet been measured for water, but I believe they will economize water.

Hindered-Settling Mill-Classifiers.

To get the hindered-settling method into practical mill form, one must provide for each compartment:

1, means of feeding it with pulp. See *B*, Fig. 25.

2, a teeter-chamber or hindered-settling chamber, *H*, in which the separation takes place.

3, a constriction at the bottom, *C*, of the teeter-chamber, down through which the heavy product may or may not go, according to the design.

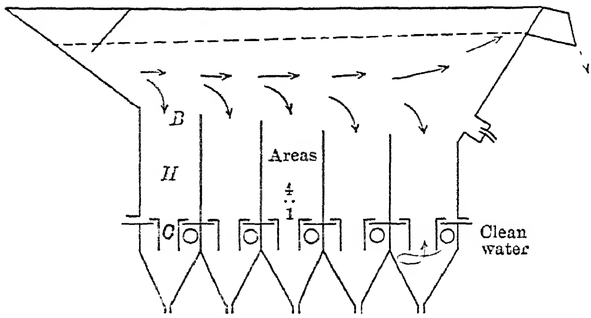
4, means of removing the product by overflow-discharge or spigot.

To put the hindered-settling principle into practical final form I have made five designs:

No. 1 hindered-settling vortex-classifier, Fig. 25.
No. 2 hindered-settling vortex-classifier, Fig. 26.
No. 3 hindered-settling vortex-classifier, Fig. 27.
The hindered-settling inverted pulsator-classifier, Fig. 28.
The hindered-settling direct pulsator-classifier, Fig. 29.

Taking these up in order:

No. 1 Hindered-Settling Vortex-Classifier, Fig. 25, shows at a glance its relation to the hindered-settling test-tube, see Figs. 16 and 18. Here one sees the hindered-settling chamber, *H*, and below it the constriction, *C*, for each of the pockets, and again below the constriction and pressure-box furnishing the upward current in a helical form and also the water for the spigot. This classifier submits the sand to hindered-settling treatment in the hindered-settling chamber, the grains there being in the condition of teeter, the heavy grains quickly going down and out, and the medium grains, remaining at full teeter, act as a filter for filtering out the fine slime. Fine free mineral is scarcely discernible in the tailings of Wilfley tables treating these products.



Observer.	Areas Ratio. Coarse Column.	Fine Column.
Schumacher,	4.7	12
Manahan,	3.3	8
Tucker,	2.5	4

FIG. 25.—No. 1 HINDERED-SETTLING CLASSIFIER.

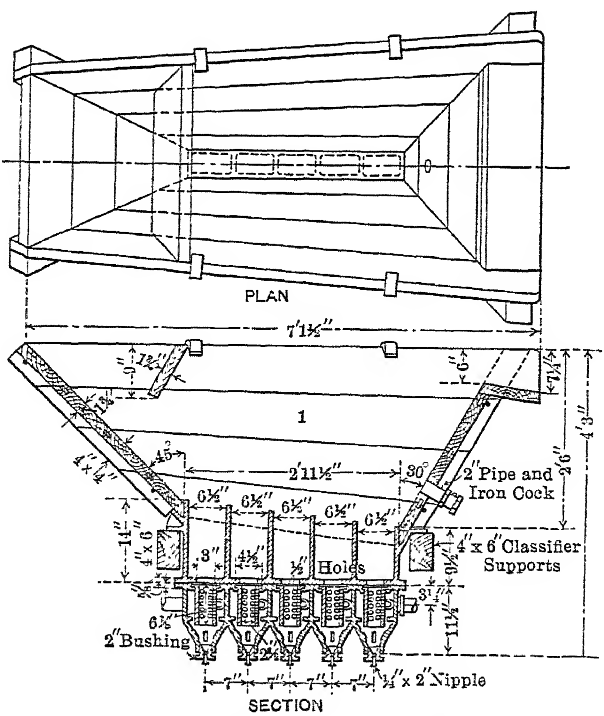


FIG. 26.—No. 2 HINDERED-SETTLING CLASSIFIER.

No. 2 Hindered-Settling Vortex-Classifier, Fig. 26, has constriction considerably longer than in the No. 1, and it has the additional feature of a lot of little vortex jets (20 in all) for giving a whirling current to the water in the constriction. This makes it still more impossible for the constriction to have a downward current on one side with upward current on the other, carrying slimes into the earlier spigots, and thereby guarantees tailings free from fine free mineral.

No. 3 Hindered-Settling Vortex-Classifier, Fig. 27, instead of a hindered-settling chamber above, and a constriction below with vortex jets in it, as in No. 2, has these two parts merged into one in conical form with inflowing jets all the way from the

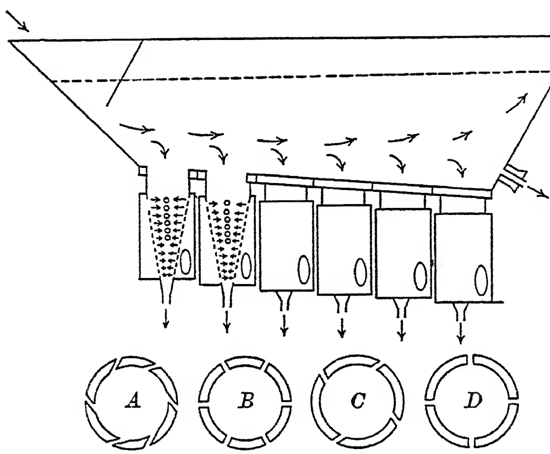


FIG. 27.—NO. 3 HINDERED-SETTLING CLASSIFIER.

top to the bottom. The inflowing jets, however, have three variations, which should be mentioned. They are in layers, alternately radial jets, *B*, *D*, and vortex jets, *A*, *C*, the vortex or tangential jets breaking up a possible sand-bank in the form of a shell around the margin, and the radial jets breaking up a possible core-bank of sand in the center. Further, the upper holes, *C*, *D*, are smaller in size, 0.25 in. in diameter, more or less, and only four to the circle or layer, whereas the lower holes, *A*, *B*, are larger in size, 0.5 in., more or less, in diameter, and six of them to the circle or layer. The conical form, combined with the arrangement of jets just referred to, gives maximum rising current two-thirds of the height down, and

gives a steadily-diminishing velocity of rising current as the water rises in the cone. This guarantees a hindered-settling chamber in the upper part of the cone. This arrangement lowers the neutral plane which lies between the rising current which does the sorting and lifts out the light stuff, and the descending current which carries down the heavy grains to be discharged at the spigot. By putting this plane considerably lower, it is believed that this No. 3 classifier is more perfect than the No. 2 from the theoretical point of each, and by having layers of jets made alternately radial and tangential, it is believed that the elimination of possible harmful currents is thoroughly accomplished, and that tailings made from treating these products will be free from fine free mineral.

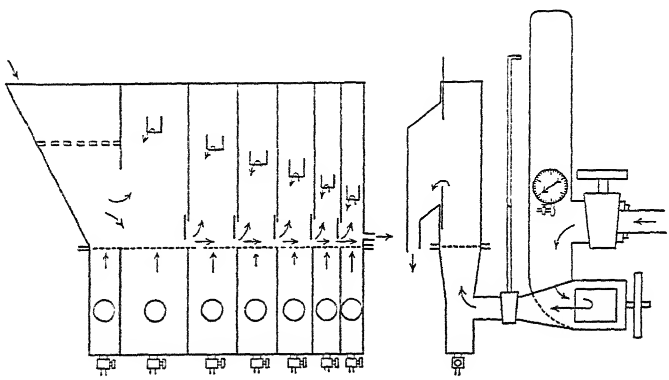


FIG. 28.—PULSATOR CLASSIFIER, INVERTED TYPE.

The Hindered-Settling Inverted Pulsator-Classifier, Fig. 28, is another mode of preventing hurtful downward currents. It is believed that oft-repeated pulsations will give an even rising current when passing up through a mass of sand; first, because it comes by pulses, and secondly, because it comes through the sieve at high velocity compared with full teeter. If the meshes of the sieve have one-quarter the area, then the velocity when passing the sieve is four times that in the space above. The up-coming water attacks all parts alike through the meshes of the sieve, which act as the constriction. This classifier, Fig. 28, has been designed to make use of this principle. It will be seen from the figure that the feed-water and sand enter the apparatus in the feed-hopper, and that the feed-hopper has a pulsa-

ting bottom to prevent the sand from solidifying and forming an obstruction on the bottom. The pulsation is given by the pulsating rising current up through the sieve in the feed-hopper as in all the succeeding compartments. The liquefied sand then passes into the first compartment and the pulsating rising current lifts out all of the lightest grains of slime and sand as overflow. The first two products carry all the slime and give the densest slime of any classifier known to me. All the heavier material finds it way forward by passing under the partition-gate into the second compartment, receiving its force to do so by the diminished height of overflow, and, therefore, head of water in the second compartment as compared with the first. The second compartment again repeats the act of the first in lifting out the lighter parts by the pulsating rising current, allowing the heavy parts to flow into the third by the diminished head of the third as compared with the second, and so on to the fourth, fifth, and sixth compartments. A safety-spigot in the end just above the sieve of the sixth compartment may be used in case an excessively heavy bank forms in that compartment. This gives a seventh product. Each compartment of the series discharges a heavier product than the one before it, and each furnishes a very perfectly classified product. The slimes that come in with the feed are taken out by the first and second compartments and appear in these two products. The Wilfley tables which treat these two slimy products cannot make tailings free from fine free mineral with one treatment, as all the other Wilfleys can. They require an additional treatment. The tailings of the Wilfley tables treating the third and fourth, fifth and sixth spigots of this classifier, are as clean and free from fine free mineral as it is likely will ever be obtained in mill practice. One may say that they are commercially free from fine free mineral. The fifth and sixth products may yield tailings on jig or Wilfley table, with sufficient amount of included grains to make their further treatment by recrushing and re-treatment necessary. The latest form of revolving-cock for getting the pulsating current is designed with the feed-water entering the end of the hollow cylindrical revolving part and discharging through two ports on opposite sides. This makes it a perfectly balanced valve and prevents the valve from pressing too hard on the valve-seat. This alternate

opening and closing of a revolving-cock gives an alternating condition of rising current and repose, repeated perhaps at the rate of 200 times a minute, and produces the desired result. By using a single revolving-cock and distributing the current through seven regulating-cocks, the seven compartments can be furnished each with its own pulsating rising current. The air-chamber used with this apparatus differs from the air-chambers ordinarily used with pumps and other hydraulic apparatus, in that the latter are used simply to diminish the shock due to the water-hammer, while this air-chamber is used mainly for the purpose of giving a sudden impulse to the water the instant the revolving-cock is open. This impulse is due to the elasticity of the air, and without it the classifier could not be

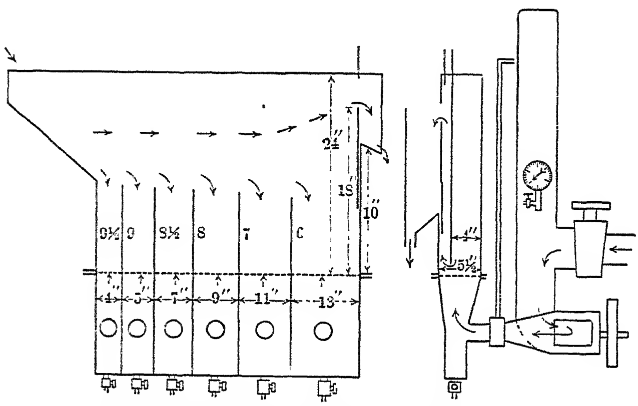


Fig. 29.—Pulsator-Classifier, Direct Type.

operated. The avoidance of shock is necessary with the classifier, as it is with all hydraulic apparatus, but it is not the main feature for which the air-chamber is used. For the method of cleaning up the first and second products of this classifier, see the description of the treatment of the slimes of the direct pulsator-classifier. The method in both cases is the same.

The Hindered-Settling Direct Pulsator-Classifier, Fig. 29, differs from the inverted only in the fact that the pockets are not connected at the bottom of the dividing partitions, and in consequence the earlier pockets overflow into the later within the machine, instead of overflowing as separate products. All the compartments receive pulsating rising current; and as a result the first compartment gives the heaviest largest grains

and the final overflow gives the lightest. The several compartments discharge their products either by the gate-and-dam discharge, commonly called the cup-discharge, used in jigs, or by the ordinary plug-spigot, with or without a shield behind it, beneath which the discharge sand must come. The ordinary spigot is the simpler and is very good. It yields its series of products graded from coarse to fine. It is not difficult with this classifier to bring the discharge of the first compartment up to a quality of finished concentrates, and in this way eliminate one jig from the set of finishing-machines. If the first be made a finished product the second can go to the Harz jig, and it will be found to give tailings free from fine free mineral. These tailings are free from fine free mineral, but they may have included grains, and require recrushing and re-treating in consequence. The succeeding spigots, third, fourth, fifth, and sixth, can all give tailings free from fine free mineral on Wilfley tables. This fulfills the requirements of milling, leaving only the slime as the single product to be finished, and this can be finished by running it upon the Wilfley table, making tailings containing fine free mineral, which are finished by Callow screen, sending the undersize to a vanner. The oversize of the Callow screen and the tailings of the vanner will both be free from fine free mineral. The slime from the Wilfley table will have to be treated by vanners or by any of the slime-methods best approved in the district under consideration.

The Hindered-Settling Pulsator Pebble Sand-Slime Cone, illustrated in Fig. 30, has been tried on a small scale and has shown remarkable abilities for separating sand and slime. By using pebbles, perhaps 1 in. in size, supported on a very coarse screen, a pulsating current furnishes a very perfect separation of sand from slime. When one realizes that the rising current in the interstices is about four times what it is in the space above, one sees that there is great opportunity for regulation of the rising current and for the avoidance of downward currents carrying slime into the sand.

The Work of the Hindered-Settling Classifier.

Fig. 31 represents the work of a hindered-settling classifier. The side tube represents here the relations between the grains

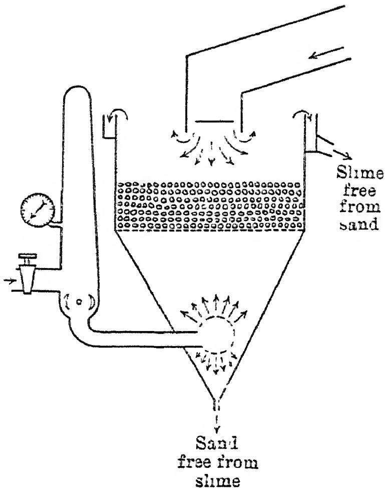


FIG. 30.—PULSATOR PEBBLE-CONE SAND-SLIME SEPARATOR.

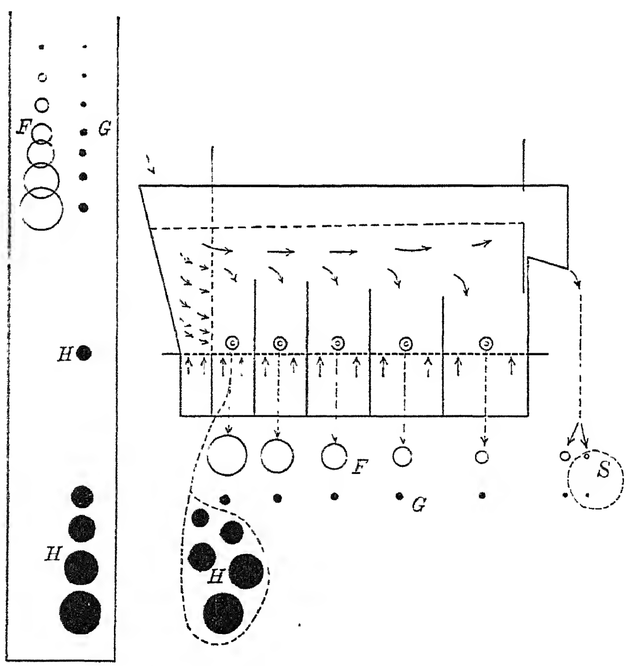


FIG. 31.—WORK OF DIRECT PULSATOR HINDERED-SETTLING CLASSIFIER.

of quartz and galena under hindered-settling condition, namely, the quartz grains in juxtaposition with the galena grains are 6.9 times the diameter of the latter. Laying this diagram down under the direct pulsator-classifier, we have an illustration of the products made by a hindered-settling classifier. Just as the free-settling classifier had an added increment, *H*, of heavy mineral in the coarsest spigot, truly-classified products in all the between spigots, and an added increment in the finest product of slimes, so has this hindered-settling classifier. It, however, differs from the free-settling classifier in having a very much larger ratio between the diameters of the grains in juxtaposition in the between products, namely, the quartz grains are 6.9 times the diameter of the galena grains associated with them, and as a consequence the coarsest product has a much larger proportion of all the values in it than was the case in the free-settling classifier. The amount of water in the spigots and in the overflow is much reduced from that in the free-settling classifier. The fine free mineral grains liable to contaminate the tailings of the concentrators that follow are much easier kept out from those tailings.

What I have said here of the direct pulsator-classifier is in the main true of hindered-settling classifiers No. 2 and No. 3 and also of the inverted pulsator-classifier. The latter does not reduce the water in the spigots so much, but reduces the water in the overflow more.

Mill-Concentrators, Some of Their Faults, and the Benefits Derived from Sizing or Classification.

The Harz Jig.—Fig. 32 represents the Harz jig. We come now to the study of the effects of improved classification upon the different machines, and we will first take up the Harz jig, and call attention to the fact that this jig has a plunger which falls and rises, and as it does so, it first sends a pulsion of water up through the grains of sand, causing the same action as the hindered-settling classifier, and, second, the water goes down through the bed of sand, sucking down into the hutch the fine grains which are lying in the interstices of the quartz.

Fig. 33 represents the curve of a Harz jig where the movement down and up of the plunger is by simple eccentric, and these curves represent in vertical direction the distance up and

down of the movement, and in horizontal direction the time elapsing while the movement is being made. The curve marked *P* represents the movement of the plunger: the curve marked *W* represents the movement of the water: *R*, the movement of the rock or quartz, and *O*, the movement of the ore or heavy mineral. The describing pencil moved from left to right, as indicated by the arrows, so that the left-hand side of the curves represents the pulsion or rising current of water, and the right-hand side represents the suction or downward rush of water.

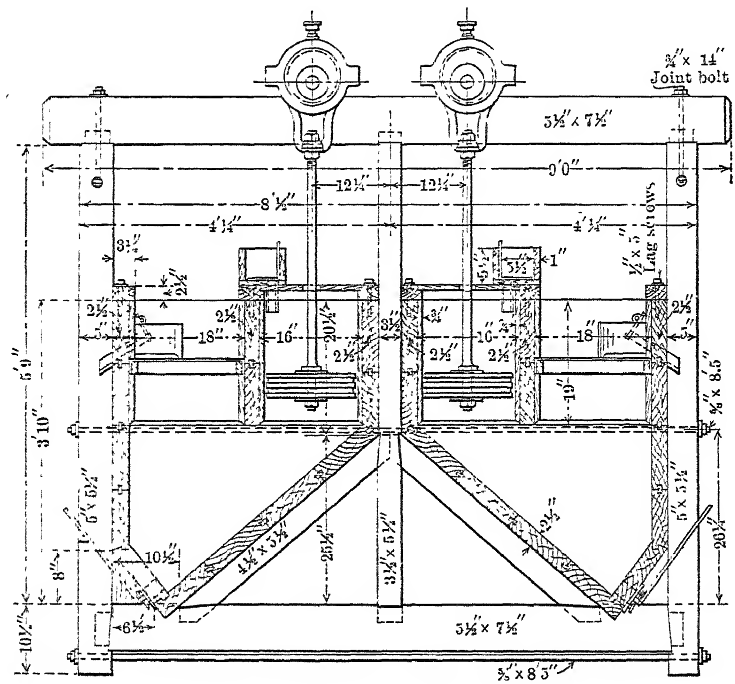


FIG. 32.—HARZ JIG. CROSS-SECTION.

Jigging Mixed Feed.—Fig. 34 shows points of strength and of weakness in jigging mixed sizes upon a Harz jig. It can yield a large quantity of good concentrates in the side-discharge, it can yield a large quantity of clean hutch-work in the grains which pass down through the sieve, but it fails in the fact that the tailings carry a large amount of fine free mineral which is too fine, and therefore too high up in the bed, to be drawn down into the hutch by the suction of the plunger.

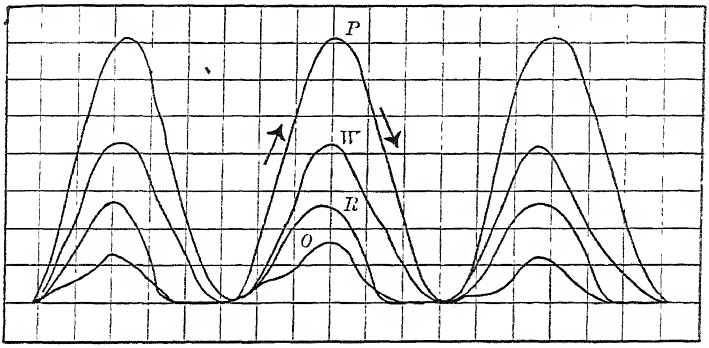


FIG. 33.—HARZ-JIG CURVES.

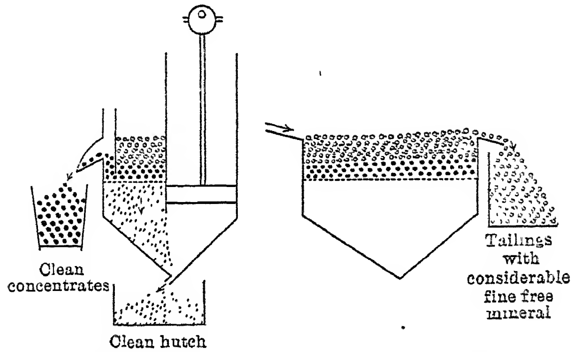


FIG. 34.—WHY MIXED FEED TO JIG NEEDS EITHER SIZING OR CLASSIFYING.

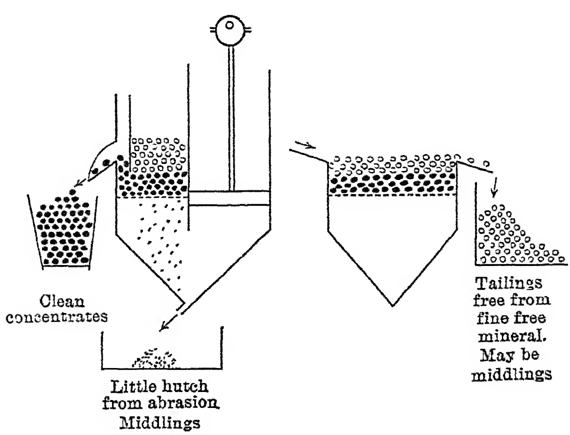


FIG. 35.—WHAT SIZED FEED DOES TO IMPROVE JIGGING.

Jigging Sized Feed.—Fig. 35 shows the improvement in jigging that is brought about by jigging a sized product. Here the side-discharge is clean concentrates. The amount of hutch is extremely small, and consists of particles abraded from the ore

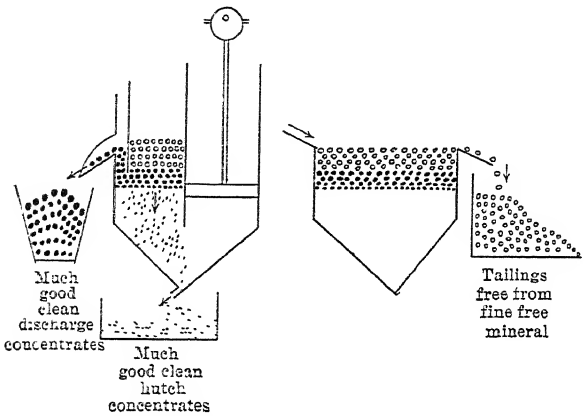


FIG. 36.—WHAT THE FIRST SPIGOT OF CLASSIFIER DOES TO IMPROVE JIGGING.

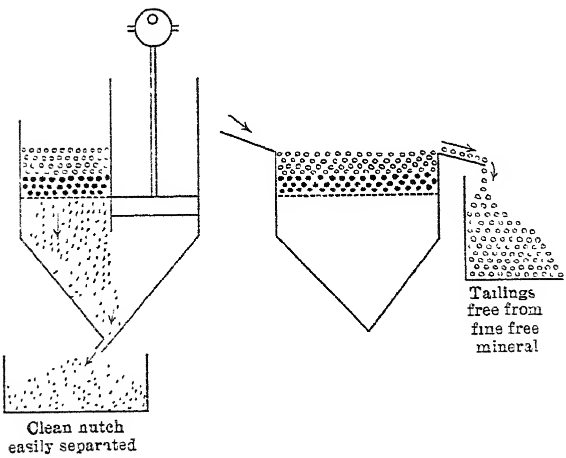


FIG. 37.—WHAT THE SECOND SPIGOT OF CLASSIFIER DOES TO IMPROVE JIGGING.

on the jig-bed. The tailings are free from fine free mineral. They probably have included grains, and if so may require re-crushing, but the removal of fine free mineral from the tailings places the jig in the most favorable aspect.

Jigging First-Spigot Product of Classifier.—Fig. 36 shows the

improvement that may be had by jigging the first-spigot product of the classifier. Here the side-discharge concentrates and the hutch are both clean and very large in quantity, and the tailings are free from fine free mineral, because, while a certain amount of small grains of heavy mineral finds its way into the first spigot, the grains are not small enough to go into the tailings as fine free mineral.

Jigging Second-Spigot Product.—Fig. 37 shows improvement due to second spigot of the classifier. Here the coarse free mineral is all wanting, and as a result, we simply make hutch-work and considerable of it, and this hutch-work will be clean concentrates. This jig will require a bed of coarse heavy mineral added to it. The tailings will be free from fine free mineral, because, while the grains of heavy mineral are much smaller than the grains of quartz, they are not fine enough to go into the tailings as fine free mineral.

Free-Settling and Hindered-Settling for Jig-Feed Compared.

Why are the two coarsest spigot-products of a hindered-settling classifier better for jig-feed than the two coarsest spigot-products of a free-settling classifier? *Answer.* The coarsest spigot-product is much richer, and in consequence the jig has less quartz to separate from it, and a larger proportion of heavy mineral is saved from the coarsest product. The next to the coarsest spigot has already had removed from it all the concentrates which go into the automatic side-discharge (cup-discharge), and the only work left for the jig to do is the suction-work of putting the small grains of concentrates into the hutch. For this work a bed of coarse mineral will have to be placed on the jig-screen, to prevent the quartz from being sucked down into the hutch. Spigot-products finer than the two coarsest will ordinarily go to the Wilfley table. If the third in size has to go to a jig, what was said of the next to the coarsest will be true of the third also.

Why is the second- or third-spigot product of a hindered-settling classifier better feed for a plunger-jig than the second- or third-spigot product of a free-settling classifier? *Answer.* In the paper, Cycle of the Plunger Jig,[9] it is shown that rapid

[9] *Trans.*, xxvi., 3 to 32 (1896).

suction and rapid jigging take place when the ratio of diameters of quartz and heavy mineral is 3.52 or larger. It is also shown that slow suction and slow jigging take place when the ratio of diameters is 2.61 or less.

Taking from the paper, Close Sizing Before Jigging,[10] the free-settling ratios for a fall of 9 in. per second, or 228.6 mm. a second, and the hindered-settling ratios as obtained from the average of the three columns computed, we have Table XII.

TABLE XII.—*Free-Settling and Hindered-Settling Ratios of Various Minerals.*

	Free-Settling Ratios for 228.6 mm. a Second, Fastest Grains.	Hindered-Settling Ratios.
Copper............	3.75	8.598
Galena............	3.75	5.842
Wolframite......	3.26	5.155
Antimony.........	3.00	4.897
Cassiterite.........	3.12	4.698
Arsenopyrite......	2.94	3.737
Chalcocite.........	2.17	3.115
Pyrrhotite.........	2.08	2.808
Blende............	1.56	2.127
Epidote...........	1.46	2.037
Anthracite........	......	5.611

Assuming that 3.52 is the dividing-line between easy jigging and difficult jigging with much suction, then we have: free-settling only includes copper and galena as having the favorable ratios, while hindered-settling includes copper to arsenopyrite and all the minerals between as having a ratio favorable to suction.

Whether the jig is fed with products from free-settling classifier or hindered-settling classifier, the tailings will be free from fine free mineral, if the classifier is a good classifier and is properly run.

HOW TO IMPROVE FINE JIG-TAILINGS WHICH HAVE RESULTED FROM MIXED FEED.

The loss is mostly fine free mineral, and a fine screen will put all this into the undersize, which can go to Wilfley table.

[10] *Trans.*, xxiv., 409 to 486 (1894).

The oversize will be clean waste up to the limit possible. If it has too much included grains it will need to be recrushed.

The Pulsator-Jig.—Fig. 38 represents the plan, longitudinal and cross-sections of the pulsator-jig. Here the pulsations are given to the sand from the forward current due to the opening and closing of the cock, which revolves 100 times a minute, more or less, giving about 200 forward pulsions, with an instant of repose between the pulsions. Obviously the hutch-product is somewhat reduced in this jig, and on that account its main field lies in jigging sized products. If there

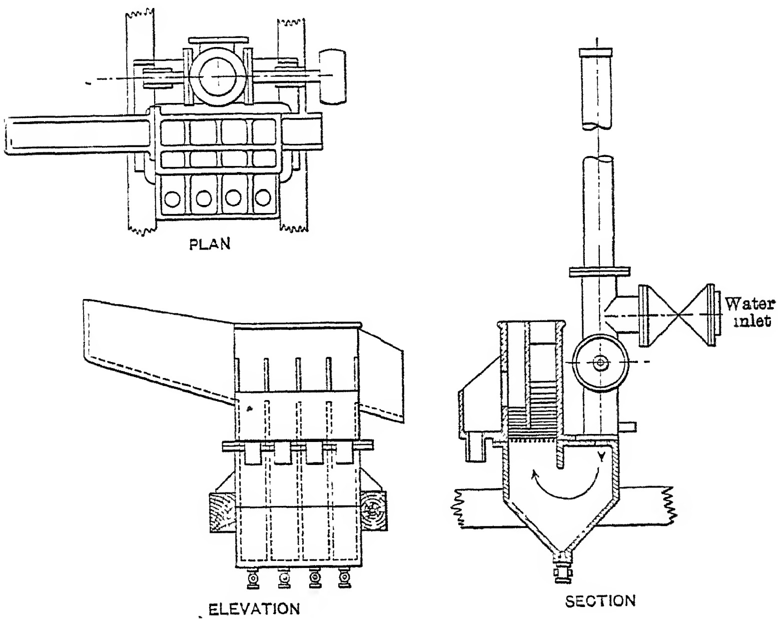
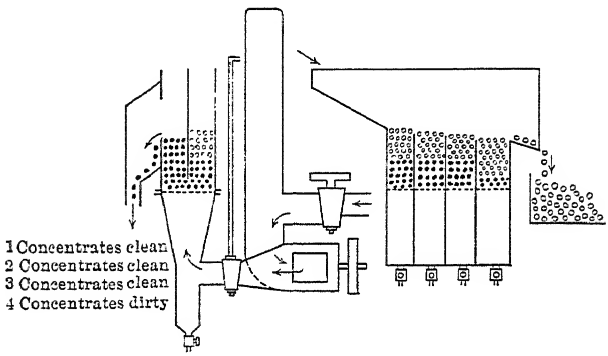


FIG. 38.—THE PULSATOR-JIG.

are small crumblings coming off from the concentrates, due to abrasion, these small grains may find their way into the tailings. If it is used as a machine for treating the first-spigot product of the classifier, some of the finest grains in this classifier-product may find their way into the tailings. This is a disadvantage of the pulsator-jig. It is, however, completely overcome by screening the tailings, in which case the tailings-oversize is turned out just as clean as the best Harz-jig tailings, while the undersize goes back to classifier and to Wilfley table. The ad-

vantages gained by the pulsator-jig far more than compensate for the above-mentioned disadvantage. By omitting the suction, the matting together of the sands in the bed of the jig, which is so marked in the Harz jig, is avoided, and as a result the jig has a very large capacity for doing work. The mistake one is liable to make, in starting a pulsator-jig for the first time, is to cut down the added "hydraulic" water so as to make the upper quartz layer about like that on a Harz jig. When this is done the bed of heavy mineral below is very hard and immovable. To correct this the quartz bed must be so completely liquefied that the bed of heavy mineral is at full teeter below. When this has been done the great capacity of



Consumes one-twentieth the power; uses about one-half the water; occupies about one-fifth the space. Makes a little fine free mineral in the tailings, which is saved in recrushing and re-treatment or by screening.

FIG. 39.—WHAT THE PULSATOR-JIG DOES TO IMPROVE JIGGING.

the pulsator-jig will be at once available, and its ability to control its discharges will also be available. Summing up the advantages of the pulsator-jig over the Harz jig, the former requires 5 per cent. or less power for driving, it requires perhaps one-half the water, it requires less than one-fifth the space in the mill, and the only disadvantage incurred in order to gain these improvements is that it may be necessary to screen the tailings.

Fig. 39 represents a pulsator-jig making its discharge, concentrates and middlings, and making its tailings with a small quantity of fine free mineral.

Comparing the pulsator-jig with the plunger-jig as they have been tested in the mills:

At Great Falls, Mont., a 4-compartment pulsator-jig with compartments 4 in. square displaced 8 plunger-jigs with 2 compartments each, and with compartments 19.5 in. wide and 39.5 in. long. The plunger-jigs had 12,324 sq. in. of sieve, while the pulsator-jig had 64 sq. in., or 1 sq. in. of sieve in the pulsator-jig did the same work as 192 sq. in. in the plunger-jig. The pulsator-jig tailings were slightly higher in copper, but the plan of screening them to remove the small quantity of fine free mineral had not then been devised.

At Britannia Beach, British Columbia, one 4-in. 6-compartment pulsator-jig has replaced five 2-compartment plunger-jigs with compartments 24 by 48 in. The total plunger-jig screen-area was 11,520 sq. in. The total pulsator-jig screen-area was 96 sq. in. In this case 1 sq. in. of sieve-cloth did the work of 120 sq. in. of sieve in the plunger-jig. The results are reported as being quite as good with the pulsator- as with the plunger-jig.

Feed for Wilfley Table.

The Wilfley table may be fed with: (1) mixed feed or natural feed, that is, unclassified and unsized stuff just as it comes from the crushing-machine; (2) sized feed which has been screened on 20-mesh, 30-mesh, 50-mesh, 80-mesh, and through 80-mesh, or some other suitable series of screens; (3) classified feed prepared by free-settling or hindered-settling classifier.

Table XIII.—*Complete Sizing-Assay Test of Wilfley Run on Natural Feed. 2 mm. to 0.*

Size in Millimeters.		Concentrates.			Middlings.			Tailings.			Slimes.		
		Weight. Tons.	Assay Per Cent.		Weight. Tons.	Assay Per Cent.		Weight. Tons.	Assay Per Cent.		Weight. Tons.	Assay Per Cent.	
Through.	On.		Galena.	Quartz.		Galena.	Quartz.		Galena.	Quartz.		Galena.	Quartz.
	2.06	0.017	100.00	0.00	0.132	73.47	26.53	3.777	0.40	99.60	......	......	......
2 06	1 63	0.059	99.77	0.23	0.795	75.39	24.61	14.713	0.40	99.60	......	......	......
1.63	1.44	0.066	99.80	0.20	0.783	69.47	30.53	11.145	0.50	99.50	......	......	......
1.44	1 27	0.112	99 88	0.12	1.202	62.80	37.20	8 085	0.20	99.80	......	......	......
1.27	1.10	0.084	99.76	0.24	0.954	54.75	45.25	6.174	0 01	99.99	......	......	......
1.10	0 97	0 120	99.43	0.57	1.056	43.80	56.20	7.322	0.02	99.98	......	......	......
0.97	0.84	0.174	98.42	1.58	1.552	31.40	68.60	5.191	0.00	100.00	......	......	......
0 84	0.68	0.206	96 54	3.46	2.021	20.10	79.90	3.350	0.00	100.00	......	......	......
0.68	0.57	0.245	94.47	5.53	1.865	16.70	83.30	2.078	0.00	100.00	......	......	......
0.57	0.45	0.506	91.05	8.95	3.112	11.30	88.70	2.501	0 00	100 00	......	......	......
0.45	0.36	0.402	83.81	16.19	2.326	8 11	91.89	1.581	0.08	99.92	0.156	0.00	100.00
0.36	0.28	0.445	81.24	18.76	2.526	6.44	93.56	1.056	0.12	99 88	0.047	10.20	89 80
0.28	0.24	0.306	81.01	18.99	0.848	4.92	95.08	0.523	0.69	99.31	0.036	10.00	90.00
0.24	0.20	0.375	84.39	15.61	1.230	3.68	96.32	0.633	1 24	98 76	0.053	10.81	89.19
0.20	0.15	0.307	89.48	10.52	0 648	4.44	95.56	0.739	1.45	98.55	0.063	12.58	87.42
0.15	0.12	0.209	95.08	4.92	0.207	6.65	93.35	1.139	1.82	98.18	0.052	19.00	81.00
0.12	0.10	0.110	95.08	4.92	0.075	7.26	92.74	0.624	2.88	97 12	0.068	24.28	75.72
0.10	0.08	0.189	97.74	2.26	0.061	32.23	67.77	0.948	3.43	96.57	0.134	27.04	72.96
0.08	0.00	0.182	99.45	0.55	0.093	90.46	9.54	1.277	7.50	99.49	0.935	28.31	71.69
Total Tons ..		4.114	90.08	9.92	21.486	23.58	76.42	72.856	0.51	82.50	1.544	18.90	81.10

TABLE XIV.—*Complete Sizing-Assay Test of Wilfley Run on Classified Feed.*

Size in Millimeters.		Sizing-Test of Feed			Concentrates			Middlings.			Tailings		
		Per Cent				Assay Per Cent			Assay Per Cent.			Assay Per Cent	
Through.	On	Quartz.	Galena.	Middlings.	Weight Tons	Galena	Quartz	Weight Tons	Galena.	Quartz.	Weight Tons	Galena	Quartz
2.83	2.49	..	.	...	.	.	.	.	.	....	0.043	0.00	100.00
2.49	2.06	1.517	.	.			.	0.001	0.00	100.00	1.336	0.00	100.00
2.06	1.63	20.299	.	0.006	0.001	100.00	0.00	0.015	8.84	91.16	18.429	0.08	99.92
1.63	1.44	20.865	0.004	0.009	0.001	100.00	0.00	0.026	10.87	89.13	21.674	0.07	99.93
1.44	1.27	19.856	0.003	0.013	0.002	93.10	6.90	0.037	21.33	78.67	20.207	0.48	99.52
1.27	1.10	15.746	0.004	0.016	0.002	91.45	8.55	0.014	28.29	71.71	11.066	0.09	99.91
1.10	0.97	5.161	0.006	0.011	0.002	89.69	10.31	0.018	39.39	60.61	7.877	0.18	99.82
0.97	0.84	5.925	0.027	0.023	0.011	95.63	4.37	0.078	38.92	61.08	7.835	0.18	99.82
0.84	0.68	2.734	0.063	0.017	0.036	96.15	3.85	0.120	29.42	70.58	2.676	0.54	99.46
0.68	0.57	1.527	0.147	0.013	0.124	97.49	2.51	0.194	25.14	74.86	2.098	1.16	98.84
0.57	0.45	1.558	0.865	...	0.730	98.01	1.99	0.494	31.00	69.00	1.122	6.04	93.96
0.45	0.36	0.217	0.678	...	1.208	98.80	1.20	0.188	47.93	52.07	0.121	16.00	84.00
0.36	0.28	0.107	1.215	...	0.687	98.24	1.76	0.089	49.53	50.47	0.097	33.05	66.95
0.28	0.24	0.028	0.517	....	0.384	98.96	1.04	0.014	46.20	53.80	0.038	27.60	72.40
0.24	0.20	0.015	0.414	.....	0.393	99.35	0.65	0.011	52.99	47.01	0.024	19.00	81.00
0.20	0.15	0.009	0.233	..	0.223	99.58	0.42	0.003	71.90	28.10	0.019	13.32	86.68
0.15	0.12	0.005	0.107	....	0.084	99.81	0.19	0.002	88.22	11.78	0.014	15.30	84.70
0.12	0.10	0.003	0.025	.. .	0.021	99.82	0.18	0.001	92.04	7.96	0.009	40.40	59.60
0.10	0.08	0.005	0.029	....	0.029	99.83	0.17	0.001	91.95	8.05	0.029	55.55	44.45
0.08	0.00	0.009	0.023	....	0.912	99.64	0.36	0.001	96.40	3.60	0.029	87.00	13.00
Total Tons		95.526	4.370	0.108	3.970	98.62	1.38	1.307	33.95	66.05	94.743	0.36	99.64

Settling Ratio 1 11

Table XIII., taken from the paper The Wilfley Table,[11] shows the results of concentrating a batch of ore with natural feed or mixed feed on a Wilfley table, and Table XIV. shows the results of feeding classified feed, second spigot of a free-settling classifier fed with from 2-mm. to 0 material, on a Wilfley table. The mixed feed gave for 100 tons of feed 4.1 tons concentrates with 90 per cent. of galena, 21.5 tons middlings with 23 per cent. of galena, and 72.8 tons tailings with 0.5 per cent. of galena. The classified feed gave 3.9 tons concentrates with 98 per cent. of galena, 1.3 tons middlings with 33 per cent. of galena, and 94.7 tons tailings with 0.36 per cent. of galena, showing a great improvement on all three lines above referred to. Tables XIII. and XIV. show many interesting facts, and a study of them will well repay the time spent.

Tse's Test.—Tse has demonstrated all the above points quantitatively. His test was to start the table, Fig. 40, working upon mixed feed of quartz and galena from 2.0 mm. to 0, and when running perfectly to slip in 30 little trays 1 in. wide in such position as to catch all the products except the finest slimes;

[11] *Trans.*, xxxviii., 556 to 580 (1908).

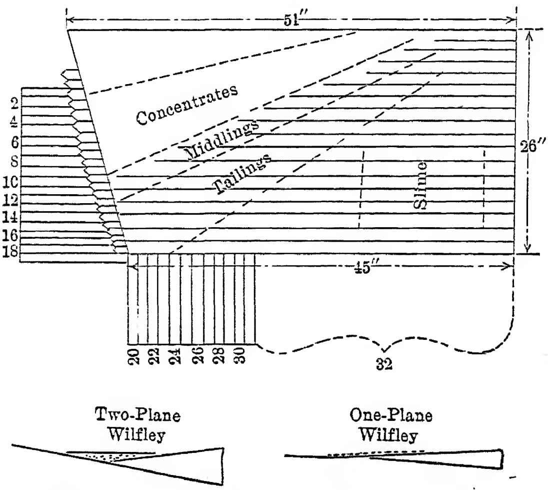


FIG. 40.—WILFLEY TABLE USED BY MR. TSE FOR TRAY-TEST.

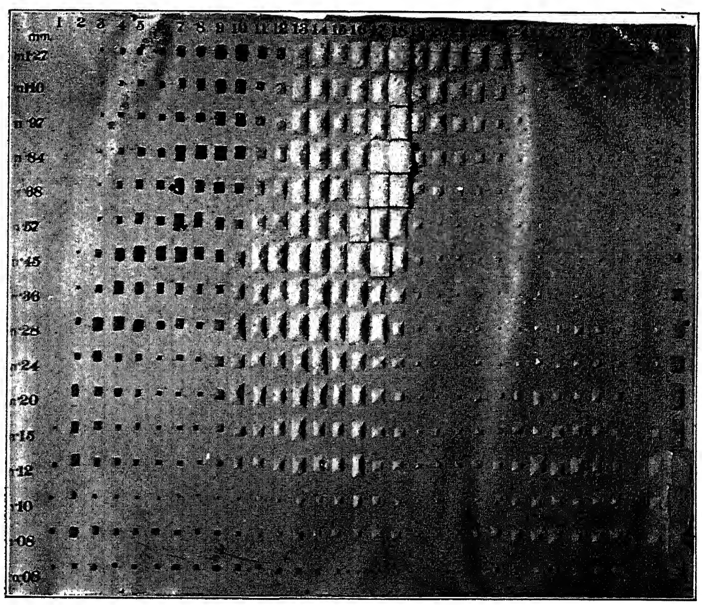


FIG. 41.—MR. TSE'S TEST.

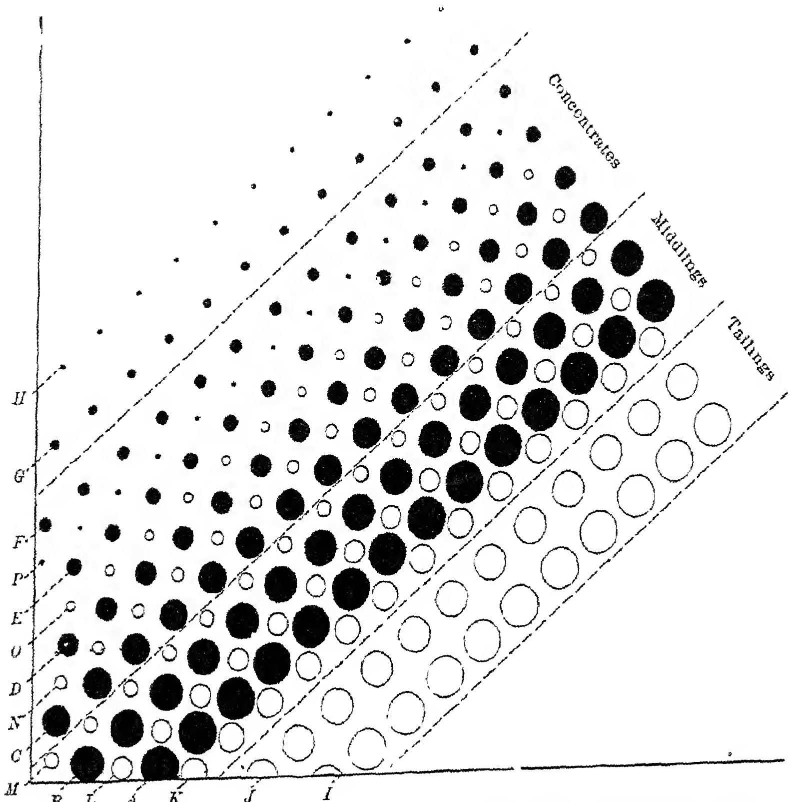


FIG. 42.—IDEAL SKETCH OF THE ARRANGEMENT OF GRAINS BY A WILFLEY TABLE.

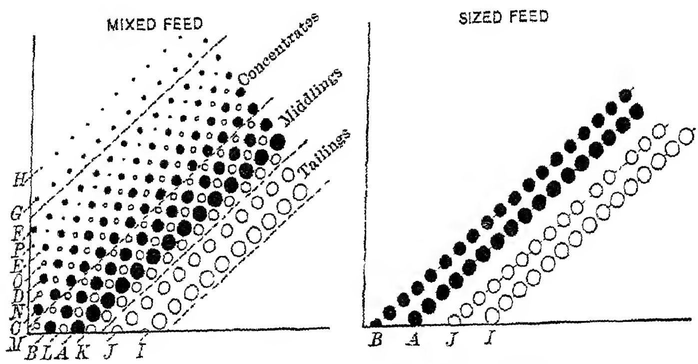


FIG. 43.—WILFLEY TABLE WITH SIZED FEED.

as soon as the trays were nearly full they were pulled out, and the time in seconds to make the catch noted. These 30 products were then sized on a series of sieves.

Trays 10, 11 and 12, Fig. 41, are the middlings. In these we can readily see the coarse galena mixed with small quartz in sufficient quantity to bury the galena very nearly out of sight. Looking at the concentrates we do not see the small quartz, but it was found in considerable quantity and was bounded by the triangle whose angles are on the three heaps defined as follows:

1, tray 9, sieve 0.28; 2, tray 5, sieve 0.20; and 3, tray 9, sieve 0.12. The fine free mineral of the tailings is partly visible on the photograph. It is shown on line through sieve 0.08, from tray 13 to tray 24 or further.

Mixed Feed for Wilfley Table.—By studying Fig. 42, which is constructed from Table XIII. and corroborated by Mr. Tse, and represents the bands of mineral grains where the table was fed with mixed feed, with the understanding that the line *H* represents the fine heavy mineral that washes over into the slime, the line *G* represents the fine free mineral that washes over into the middlings and tailings, we see that the following irregularities take place which need to be improved:

1. The concentrates are contaminated by a noticeable amount of small free quartz grains.
2. The middlings are large in quantity and they have large free mineral mixed with small free quartz and fine free mineral. Mr. Caetani has given the name "counter-classified," where the heavy mineral is in large grains and the quartz is in small grains.
3. The tailings are contaminated with fine free mineral.

How Sized Feed Helps a Wilfley Table.—In the diagram, Fig. 43, we have placed the coarse quartz and the coarse galena just where they would be found in the mixed-feed bands of Fig. 42. The actual figures obtained, calculated for 100 tons of sized feed, gave 6.5 tons concentrates with 99 per cent. of galena, 1.6 tons middlings with 59 per cent. of galena, and 91.8 tons tailings with 0.04 per cent. of galena. It is not probable that the bands will stay so far apart as shown in the figure; in fact, they will run together a little. The small lot of middlings produced was 1.6 tons with 59 per cent. of galena, and these grains are per-

fectly separable on a Wilfley table. In consequence, the middlings can go to the little raff-wheel and be sent back into the feed without any harm whatever; they will not accumulate unless included grains or middle-weight mineral is present. Then different provision will have to be made. Summing up, we have:

The concentrates are free from small quartz; the middlings are small in quantity and can be fed over again unless included grains or middle-weight mineral is present; the tailings are free from fine free mineral.

All of which is as satisfactory as possible.

How First Spigot of a Classifier Helps a Wilfley Table.—In the diagram, Fig. 44, we have placed the coarse quartz and the small galena, together with the added increment of larger

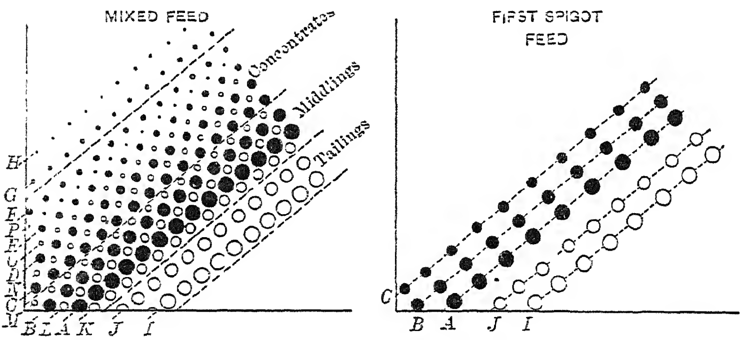


FIG. 44.—WILFLEY TABLE WITH FIRST-SPIGOT FEED.

galena, approximately where they would be found in mixed-feed bands, see Fig. 42. If actually drawn to a scale the band of galena would be much wider than is shown in this diagram. The bands will not stay so far apart, but will mix a little. In my test, the middlings ran up to 5.5 tons in 100, but that I think is unnecessarily high. These middlings, unless included grains or middle-weight mineral is present, can be returned and fed over by raff-wheel and so save further complications. The actual figures obtained and rated for 100 tons of feed were: concentrates, 49.6 tons, with 99 per cent. of galena; middlings, 5.5 tons, with 13.8 per cent. of galena; and tailings, 44.8 tons, with 0.29 per cent. of galena, all of which is a thoroughly satisfactory improvement over mixed feed. The concentrates are

free from quartz, the middlings are small in quantity, the tailings are practically free from fine free mineral.

How Free-Settling Second Spigot Helps a Wilfley Table.—Here, see Fig. 45, the bands are placed far apart, as they would be in the mixed-feed diagram. If actually drawn to scale they would be farther apart than as shown in this figure and the galena grains would be smaller relatively to the quartz. The actual figures obtained on basis of 100 tons were: concentrates, 3.9 tons, with 98.6 per cent. of galena; middlings, 1.3 tons, with 33.9 per cent. of galena; tailings, 94.7 tons, with 0.36 per cent. of galena. Here again the concentrates are practically free from small quartz, the middlings are small in quantity, and the tailings are almost free from fine free mineral.

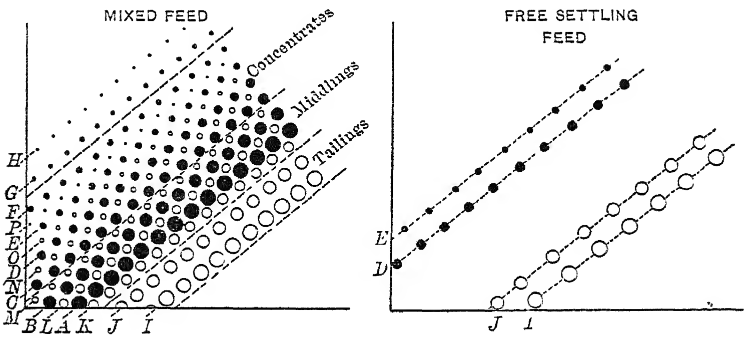


FIG. 45.—WILFLEY TABLE WITH FREE-SETTLING FEED.

If Table XIV. had been the result of hindered-settling classifier instead of free-settling classifier, the results and the above discussion would have shown still better.

How Hindered-Settling Second Spigot Helps a Wilfley Table.—Fig. 46 represents the quartz and galena bands approximately as they would occur on the mixed-feed table. They are probably not far enough apart even here if drawn to scale. In actual work these will drift towards one another and be slightly mixed in the middlings, but the middlings can be fed back as in the previous cases and disappear from the account unless they are contaminated by included grains or middle-weight mineral, in which case they must receive suitable consideration, as in the previous cases also. (This calls for another paper.) My paper on

The Wilfley Table[12] has no instance in it of hindered-settling feed. (This also calls for another paper.)

A Geometrical Suggestion.—When one watches the coarser tailings on the margin of the bands on a Wilfley table, and sees that the grains are jerked forward a little and then drop over a riffle-cleat, then jerked forward again and then drop over a riffle-cleat again, he is led to ask what is the determining feature which causes the grain to be lifted over the cleat. Fig. 47 is offered as a suggestion pointing to cause and effect. The large quartz grain has a short vertical arrow representing gravity, and a longer horizontal arrow representing the force of the water. Between these two forces lies the diagonal of the parallelogram which is the resultant of the two forces. The action would be

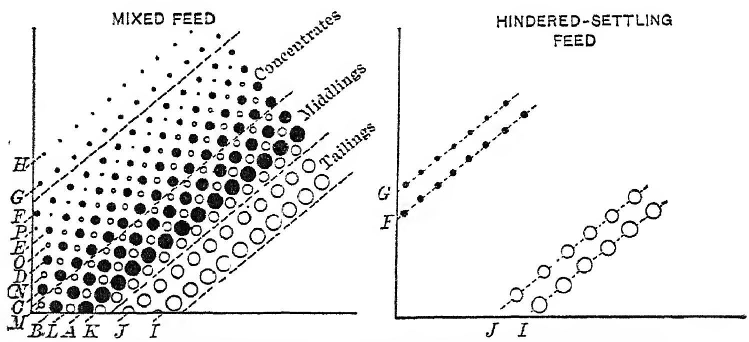
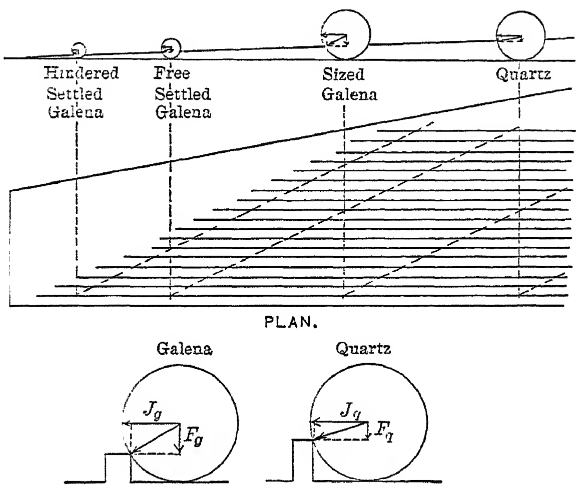


FIG. 46.—WILFLEY TABLE WITH HINDERED-SETTLING FEED.

as follows: As soon as a point on the taper riffle-cleat was reached where the resultant came above the cleat, that moment the grain would rise over the cleat and drop into the next groove. This would be repeated over and over, and thus give the path of the quartz grain down a diagonal. The sphere of galena of the same size as the quartz would have a longer vertical arrow, and therefore its diagonal would reach lower, and therefore the riffle-cleat would have to be considerably lower before the point was reached where the galena grain could rise over and drop into the next groove. This gives the large grain of galena a diagonal path considerably in advance of the quartz. Again, a galena grain related in diameter to the quartz as in the second spigot of a free-settling classifier, that

[12] *Trans.*, xxxviii., 556 to 580 (1908).

is, with a ratio of 1 to 3.5, would be far in advance again of the equal-sized grain of galena. Finally, a galena grain related in diameter to the quartz as in the second spigot of a hindered-settling classifier, that is, with a ratio of 1 to 6.9, would be still further in advance of the quartz. This cannot be regarded as a demonstration, because the grains in the figure are far too large to be true to nature, but if we consider that the vertical arrow represents the sum of all the forces that tend to keep the grain in the groove, and the horizontal arrow represents the sum of all the forces pushing the grain down the slope, then this analogy may turn out to be a true picture of fact.



J is the water-force pushing the grain across the riffles.
F is the force of gravity holding the grain in the riffle.

FIG. 47.—A STUDY OF THE ACTION OF THE WILFLEY TABLE.

What Can be Done to Improve Wilfley-Table Middlings resulting from mixed feed, supposing no preparatory classification or sizing is used? This product is "counter-classified" with grains of heavy mineral larger than the grains of quartz; see Fig. 41, trays 10, 11, and 12; also Fig. 42. This can be screened to let the small quartz and fine heavy mineral go through while the coarse heavy mineral remains on the screen as oversize; or better, it can be fed to a little free-settling classifier with one pocket. This would save more of the heavy mineral than the screen; or best of all, it could be fed to a one-

pocket hindered-settling classifier, which would save more heavy mineral than the little free-settling classifier. The rejected fine product with either method could go to a Wilfley table to finish it.

What Can be Done to Improve Tailings of Wilfley Table resulting from mixed feed with no preparatory classification or sizing? This product has included grains and fine free mineral. I am assuming that the slime has been already separated from it. This product is the exact opposite of the middlings. Free-settling classifier is better than hindered-settling, and screen-sizing is better than free-settling. The Callow screen seems to be the machine for this work. The oversize will be free

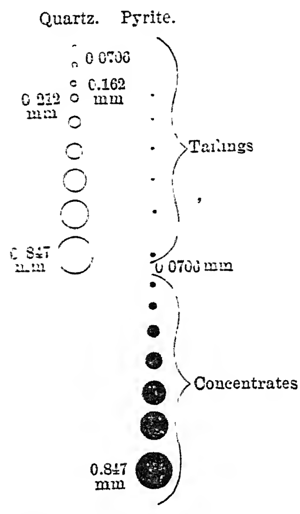


FIG. 48.—THE LOCATION OF LOSS BY A VANNER.

from fine free mineral and will in all probability be clean enough to throw away; if not, it must be crushed for included grains. The undersize can go direct to a vanner and yield clean concentrates and tailings. The slime can be sent to a vanner unless another slimer has been proved to be better.

FEED FOR VANNERS.

What Can be Done to Improve Vanner-Tailings resulting from mixed feed without preparatory classification? A vanner-feed with pulp 0.847 mm. to 0, Fig. 48, will yield in the concentrates pyrite, 0.847 to 0.0706 mm. in diameter; and in the tail-

ings, quartz, 0.847 mm. to 0, with pyrite, 0.0706 mm. to 0. This 0.0706 mm. is only the result of a single determination and needs to be corroborated, but it is believed to be about right. Here, as in Wilfley tailings, hindered-settling is not so good as free-settling, and free-settling is not so good as sizing for separating the fine from the coarse.

Hindered-settling would put into the overflow (multiplier is about 3):

Pyrite, from	0.0706 mm. to 0.
Quartz, from	0.2118 mm. to 0.

Free-settling would put into the overflow (multiplier is about 2.25):

Pyrite, from	0.0706 mm. to 0.
Quartz, from	0.1588 mm. to 0.

Screen-sizing would put into the undersize:

Pyrite, from	0.0706 mm. to 0.
Quartz, from	0.0706 mm. to 0.

Clearly the screen is theoretically the best, but the process would call for such a fine screen that it would, according to present knowledge, probably be impossible, and in consequence a free-settling classifier will probably be the best appliance to use here. The loss by included grains has not been considered, as the grains may be too fine for recrushing and redressing.

Having obtained the overflow or undersize, we next feed it upon canvas table, making clean tailings, and feed the canvas concentrate on a steep high-speed vanner, making clean concentrates, the tailings to go back to canvas.

Gaseous Decomposition-Products of Black Powder, With Special Reference to the Use of Black Powder in Coal-Mines.*

BY CLINTON M. YOUNG, E. M.,† LAWRENCE, KAN.

(Pittsburg Meeting, March, 1910.)

I. INTRODUCTION.

THE experiments herein described were carried on in 1908–9 by the State Geological Survey of Kansas. Some months before taking up work on black powder the Survey had resumed work on an interrupted investigation of explosions in coal-mines, a subject which is engaging the attention of numerous investigators. An unusual number of serious disasters has directed to the subject a study which discloses the disquieting fact that, in the United States, the ratio of lives lost per ton of coal mined has been on the increase.

Methods of coal-mining have changed to some extent with the great increase of output, attained in comparatively recent years. These changes involve the working of very extensive mines, with increased difficulty of ventilation, increased accumulation of coal-dust. and a very great increase in the amount of explosive used.

This last point assumes truly startling proportions when it is carefully observed. In the fields of the Middle West it is not now uncommon for a miner to use six kegs (150 lb.) of black powder in a "pay" of two weeks, while within the memory of men still active, one keg (25 lb.) would last two men for a "pay." So great has the use of explosives become in some fields that the skilled coal-miner is hard to find, and the coal is not mined, but blasted.

A preliminary study of the subject of explosions in coal-mines has led to the belief that the use of large quantities of explosives, in some cases by slightly-trained men, had not been

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† Associate Professor of Mining Engineering, University of Kansas, Lawrence, Kan.

estimated at its true importance as a contributing cause of mine-explosions.

While so-called "safety" or "permissible" explosives are coming into use, the explosive most used in the United States to-day is black powder, which differs from gunpowder in the use of sodium nitrate instead of potassium nitrate, with an accompanying change in the proportions of sulphur and charcoal.

The points on which it was thought that explosives might be suspected, are the possibly dangerous quality of their combustion-products, and the fact that they shatter the coal. The latter fact would not be considered a probable direct cause of mine-explosions, but it does lead to the production of much dust, which is sometimes explosive, and undoubtedly, by weakening the roof, increases the number of accidents due to falls of rock.

Unquestionably, the gases produced by the combustion of a large amount of explosive in a comparatively small room may materially change the composition of the air of the room. The statement made by many authors, that mining-powder gives off large amounts of carbon monoxide, seemed to lend importance to this consideration. A search of the literature of this subject, however, fails to reveal anything of importance. In fact, the last work of value seems to be the classical investigation of Noble and Abel,[1] who were not dealing with modern American black blasting-powder, but almost entirely with ordnance-powder. Even the little work done by them on mining-powder is comparatively unimportant to us, because the English blasting-powder differs from gunpowder in a decrease of the proportion of potassium nitrate, while in American powder a sodium salt is substituted for a potassium salt, but the proportion of oxygen is not decreased.

It became necessary, then, to examine the gaseous decomposition-products of black powder under different conditions of explosion.

Numerous samples of powder obtained from different sources, but principally from mines in the Middle West, were examined,

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[1] *Philosophical Transactions of the Royal Society*, vol. clxv., pp. 49 to 155 (1875); vol. clxxi., pp. 203 to 279 (1880).

and it was found that, while some differences exist, the powder is fairly uniform in composition.

The method of analysis was very simple. The powder was weighed as received, and again after being dried, the moisture thus being determined. The sodium nitrate was determined by solution in water, evaporation to dryness and weighing as sodium nitrate. This weight was checked by the weight of the residue. Sulphur was determined by oxidation and precipitation as barium sulphate. Charcoal was determined as the difference between the weight of the sample and the sum of the weights of moisture, sodium nitrate, and sulphur. While this method may not give results of absolute accuracy, it is sufficiently accurate for the purpose of the investigation. In some cases chlorine was determined and, calculated as sodium chloride, was found to be from 0.5 to 1 per cent. Some results of analyses are as follows:

Powder. Sample.	Moisture. Per Cent.	$NaNO_3$. Per Cent.	S Per Cent.	Charcoal. Per Cent.	Total. Per Cent.
No. 1, . . .	0.9	72.8	11.0	15.5	100.2
No. 2, . . .	0.6	71.9	11.5	15.8	99.8
No. 3, . . .	0.8	71.7	11.4	16.1	100.0
No. 4, . . .	0.7	72.1	11.1	15.9	99.8
No. 5, . . .	0.7	71.7	10.7	16.3	99.4
No. 6, . . .	0.7	71.7	10.8	16.9	100.1

These samples include three different brands of powder. Nos. 1 and 4 are of one brand, Nos. 2, 3, and 6 of another, and No. 5 of another. The samples of the same brands were obtained at different points, with the idea of ascertaining whether the powders were of uniform composition. The analyses show that the variation in the proportions of the ingredients is unimportant.

The powders examined were found to be rather highly nitrated, though it is thought that in some cases a lower proportion of nitrate may be used, thus producing a somewhat slower and cheaper powder.

In the literature of the subject the statement is frequently found that powder exploded at low pressure—that is, powder occupying only a small part of the volume of the containing vessel—produces a large proportion of carbon monoxide. In other words, when the pressure of the gases is low the proportion of carbon dioxide decreases, and that of carbon monoxide

increases. It was evident that this statement, if true, is very important, because a blown-out shot, or a shot in which the quantity of powder used is so great that the coal would yield before all of the powder had been burned, would produce a large proportion of carbon monoxide, and the gases themselves might be combustible.

This statement is evidently made upon the authority of Noble and Abel. But, as was stated before, these experimenters used English powder, and it seemed desirable to determine whether their results could be duplicated with American powder.

To test the effect of varying gas-pressures, a bomb was made in which the powder could be exploded, and from which the gases could be drawn off at will. This bomb followed quite closely the design of that used by Noble and Abel, although it was designed before the description of the apparatus used by the English experimenters had been obtained. This arrival at the same end by independent experimenters is not important to the investigation, but it is interesting.

The first bomb was made from a piece of 3-in. steel shafting, 6 in. long. A hole 0.75 in. in diameter was bored longitudinally through the center. In each end was fitted a slightly-tapered tool-steel plug, having a bearing on a shoulder, thus forming a chamber 2.96 in. long. This bearing was ground to a smooth fit with emery. In the side of the bomb, near one end, was the outlet for gases, consisting of a small hole closed with a plug having a tapered end, ground to a true fit. In this first bomb an attempt was made to make even the screw threads tight.

The charge was fired by the fusing of a small iron wire by an electric current. It was the evident difficulty of insulating a conductor passing through the wall of the bomb that led to the re-invention of the firing-device used by Noble and Abel. Through the center of one plug passed a tapered steel pin, ground to a true fit. This firing-pin was insulated with tissue-paper. At the moment of firing, the blow struck by the gases would drive this tapered pin firmly into the plug, forming a gas-tight joint. Very little trouble has been experienced with this firing-device.

It was soon found that the joint between the plugs and the

body of the bomb could not be made gas-tight, and the stream of gases escaping with great velocity and at high temperature rapidly enlarged the least leak.

It was thought that, by placing a thin copper washer between the plug and bomb, a gas-tight joint could be made, but it was found that the pressure was so great as to distort the ends of the plugs, even though there seemed to be no room for movement. Then the shoulder on the bomb was tapered slightly and a washer made to fit the space. This was found to be perfectly tight, as the washer was driven outward to a perfect fit by the gas-pressure, the joint becoming tighter as the pressure became greater. This very obtusely conical obturator seems to be as satisfactory as the very acutely conical one used at the proving grounds at Sandy Hook, N. J. Later, brass was substituted for copper as the material of the washers and, as they had an indefinite life, they were left in place in the bomb.

This first bomb failed under conditions which will be described later and which gave an unusually large volume of gas. The gas escaped, washing away so much of the material of the bomb that it became necessary to construct a new one. The second bomb was similar to the first, but was made from 4-in. shafting and had the outlet-opening in the end opposite the firing-plug. In the first bomb the outlet-opening occasionally became stopped with the solid decomposition-products of the powder, but the second bomb has given no trouble. The firing-chamber of the second bomb was also made larger than that of the first, being 1 in. in diameter and 2.7 in. long.

The volume of the gases was determined by measuring the amount of water displaced. As the gases did not bubble through the water this method was sufficiently accurate.

In order to collect gases from powder fired at very low pressures a retort was used, made of a piece of gas-pipe closed at one end. The other end was provided with a cork having two openings through which pieces of glass tubing were passed. One of these was provided with a short piece of rubber tubing, closed with a pinch-cock, through which powder was passed in small quantities. The gases passed out through the other tube. At first only a short piece of rubber tubing was used to conduct the gases to the receiver, and it was found that the explosion of a single grain of powder would give rise to a rush of

gas, followed by a back-flow which would draw water into the retort. This trouble was remedied by the use of a long piece of thin-walled tubing, partly closed by a screw-cock, which acted as an elastic receiver.

As the powder was exploded in the retort by contact with the heated end, and as this was heated only to a low temperature, it is probable that the decomposition was somewhat different from that caused by the intense heat of the fused wire and the burning of other powder-grains, which was the condition of decomposition in the bomb.

The method of analysis of gases was as follows: Carbon dioxide and hydrogen sulphide were absorbed in potassium hydroxide. In most cases hydrogen sulphide was not determined. as it was found to be rather unimportant to the investigation in hand; where not specifically mentioned it is to be understood that the results given for CO_2 include also H_2S. It was found, in the cases in which it was determined, to amount to from 3 to 6 per cent. When it was determined, a separate portion of gas was taken and passed through a solution of cadmium chloride. The CdS was filtered out, and the S determined as $BaSO_4$.

After the removal of CO_2 and H_2S, the unsaturated hydrocarbons were absorbed in fuming sulphuric acid. In the analyses these hydrocarbons are always designated as C_2H_6, no attempt having been made to determine their real composition. In powder-gases the quantity of these hydrocarbons is always small.

Next, oxygen was absorbed in potassium pyrogallate, and after this, carbon monoxide was absorbed in cuprous chloride solution. Because of the large quantities of CO sometimes found, two pipettes of the cuprous chloride solution were used, and in the later analyses, the results of explosion were so calculated as to determine CO, if the absorption should have been incomplete.

The CH_4 and H_2 were determined by combustion in an explosion-pipette, and in all of the later analyses, which include those of the most importance, the calculation was made for CO also.

The following analyses present the composition of the gases given off by the powders, the composition of which has been given before, with the exception of No. 1. In each case a charge of 10 g. was used. As the capacity of the firing-chamber

of bomb No. 1 was 22 cc., and as the specific gravity of powder is nearly 1, it is seen that the charge occupied about 45 per cent. of the volume of the chamber.

	No. 2.	No. 3.	No. 4.	No. 5.		No. 6.
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
CO_2 . . .	53.1	54.9	51.5	52.5	54.6	54.0
H_2S . . .	3.9	3.8	0.1	0.0	0.0	0.0
C_2H_6 . .	0.0	0.0	5.5	3.7	4.0	4.0
O_2 . . .	0.2	0.3	0.6	1.1	0.4	0.3
CO . . .	7.4	6.4	6.0	7.0	7.2	7.2
CH_4 . .	0.0	1.6	1.1	1.2	0.4	2.0
N_2 . . .	34.5	33.0	34.8	33.8	33.0	32.5
Total, . .	99.1	100.0	99.6	99.3	99.6	100.0
Combustible, .	11.3	11.8	12.7	11.9	11.6	13.2

II. Effect of the Presence of Coal-Dust.

As it seemed highly probable that in the presence of coal a larger proportion of combustible gases might be produced, an attempt was made to fire a mixture of coal-dust and powder. In the first experiments powder No. 7 was used, which was similar in composition to Nos. 2 to 6. An explosion of 10 g. of this powder under the conditions already described gave a gas-mixture yielding:

	Per Cent.
$CO_2 + H_2S$	58.7
C_2H_6	0.0
O_2	0.0
CO	7.2
CH_4	0.4
H_2	0.4
N_2	33.4
Total, ...	100.1
Combustible, ...	7.9

A charge of 10 g. of powder No. 7 was mixed with ten small pieces of coal from Monongah mine No. 8. It will be remembered that a very serious explosion occurred in Monongah mines No. 6 and No. 8, and coal from these mines had been obtained for examination. When this charge was exploded the bomb failed. The escaping gases washed out a considerable cavity in the side of the explosion-chamber and completely washed away the screw-threads on one side of the plug closing the gas-outlet. These facts point to a greatly increased gas-pressure, as the bomb had confined without failure the gases from the same amount of powder fired without coal.

Bomb No. 2 was then constructed and was used for all the subsequent experiments with a bomb. The capacity of the explosion-chamber was approximately 50 cc. A charge of 8 g. was used, occupying approximately 16 per cent. of the volume of the chamber, this charge having been adopted for the first experiments with this bomb. It was found later that a larger charge, of perhaps 15 or 16 g., would have been more satisfactory, as with the small charge the firing-pin was not always driven to a firm contact with the plug, and a small leakage of gas sometimes occurred, but the small charge was continued in order to give uniformity to the experiments.

Table I. gives the results of the first set of experiments in which coal-dust was mixed with the powder. In these a "FF" powder was mixed with dust made from coal from the State mine at Lansing, Kan. The coal was reduced to dust and passed, in some cases through a 100-mesh sieve and in some cases through a 200-mesh sieve. An average analysis of this coal is: moisture, 8.58; volatile and combustible, 42.2; fixed carbon, 44.5; ash, 13.28 per cent. The following experiments were made with bomb No. 1 before the failure described above:

TABLE I.—*Results of Exploding a Mixture of Coal-Dust and Powder.*

(First Set of Experiments.)

Test No.	1.	2.	3.	4.	5.	6.	7.	8	9
Powder, g,	10.0	10.0	10.0	10.0	10.0	10.0	10.0	10.0	8.0
Coal, g.,	0.0	0.02	0.05	0.1	0.25	0.5	1.0	2.0	4.0
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
CO_2	49.1	50.0	49.0	47.1	42.9	32.4	40.8	34.6	24.3
H_2S	4.8	4.4	5.2	4.1	3.1	16.9[a]	4.6	5.2	7.1
C_2H_6	0.2	0.1	0.0	0.0	0.3	1.5	0.2	0.5	1.2
O_2	0.4	0.2	0.3	0.2	1.1	0.1	0.5	0.2	0.0
CO	12.1	12.7	12.3	15.2	17.6	17.5	18.1	21.6	26.0
CH_4	0.2	0.1	1.4	1.8	1.2	2.5	4.9	10.0	17.6
H_2	2.4	2.5	0.7	0.0	0.0	3.0	0.4	0.0	0.0
N_2	30.5	31.5	31.2	30.7	33.7	26.3	30.2	27.9	24.5
Total,	99.7	101.5	100.1	99.1	99.9	100.2	99.7	100.0	100.7
Coal, per cent.,	0.0	0.2	0.5	1.0	2.4	4.8	9.1	16.7	33.3
Combustible, per cent.,	19.7	19.8	22.2	21.1	22.2	41.4	28.2	37.3	51.9

[a] This analysis shows an unusually high percentage of H_2S. As no other experiment showed an approach to this percentage, it would seem that the correctness of the analysis is open to question.

Table I. shows an increase in the percentage of combustible gases with the increase in the proportion of coal mixed with the powder. The combustibles appear roughly in quantities proportional to the amounts of coal used, and are probably formed in part by distillation from the coal.

It will be noticed that the powder used for these experiments produced a large amount of CO when fired alone, and that, in the presence of coal, it sometimes produced gas-mixtures so rich in combustibles as to be themselves combustible. This of course demonstrated the fact that black powder, or at least this particular powder, may, when fired in the presence of coal, produce combustible gases.

But as the powder used produced in one experiment as much as 12.1 per cent. of CO when fired alone, it was thought best to test another powder which gave a lower percentage of combustibles. To this end another series of tests was conducted with powder purchased from a local dealer, which had been found to yield only small quantities of combustibles. In this series the coal used was the same as for the first series, but all was screened through a 200-mesh sieve. In these experiments bomb No. 2 was used. In each case the volume of gas was measured and the large amount produced in some cases shows very plainly the effect of the coal.

TABLE II.—*Results of Exploding a Mixture of Coal-Dust and Powder.*

(Second Set of Experiments.)

Test No.	1.	2.	3.	4.	5.	6.	7.	8.	9.	10.	11.
Powder, g., .	8.0	7.75	7.5	7.0	6.0	5.0	4.0	7.0	6.0	7.0	8.0
Coal, g, . .	0.0	0 25	0.5	1.0	2.0	3.0	4 0	1.0	2.0	1.0	0.5
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Powder, . .	100.0	96.9	93.8	87.5	75.0	62.5	50.0	87.5	75.0	87.5	94.1
Coal, . . .	0.0	3.1	6.2	12.5	25.0	37.5	50 0	12.5	25.0	12.5	5.9
$CO_2 + H_2S$.	61.6	56.2	50.2	38.3	30.6	32.7	28.9	35.4	30.6	39.4	48.5
C_2H_6 . . .	0.3	0.2	0.1	0.4	0.5	0.9	0.5	0.1	0.7	0.7	0.3
O_2	0.4	0 2	0.3	0 2	0.4	0.4	0.3	0.5	0.8	0.2	0.2
CO . . .	2.8	10.4	17.4	28.2	33.0	30.8	35.0	31.6	32.1	27.5	17.7
CH_4 . . .	0.8	0.3	0.4	1.9	4.4	4.8	5.1	1.3	4.0	1.4	2.6
H_2 . . .	0.0	1.5	3.1	5.4	9.0	9.2	10.8	7.9	8.3	5.6	0.0
N_2 . . .	34.1	31.2	28.5	25.6	22.2	21.3	20.4	23.2	23.4	25.2	30.6
Total, . .	100.0	100.0	100.0	100.0	100.1	100.1	101.0	100.0	99.9	100.0	99.9
Combustible, per cent.,	3.9	12.4	21.0	35.9	46.9	45.7	51.4	40.9	44.9	35.2	22.6
Volume, cc.,	2,300	2,550	2,600	2,975	2,970	2,300	1,750	2,630	2,350	3,010	2,770

The results given in Tables I. and II. show an increase in the percentage of combustibles in the gases produced. It is evident that black powder, fired in the presence of coal, produces gases which are in some cases combustible. The gases were in all cases tested for combustibility and mixtures of the gases and air were tested for explosibility. These tests showed that the gases are combustible when the percentage of combustibles exceeds 20 per cent. of the volume. The explosive limits, that is, the greatest and least amounts of gas, the mixtures of which with air are explosive, were determined approximately. Of course, any of the gas-mixtures which is combustible is explosive, and the range of explosiveness becomes greater as the percentage of combustibles increases, and it is also somewhat influenced by the character of the gases.

The experiments do not show uniformity, nor do those of other experimenters. No theory which has been advanced to account for the reactions taking place in the combustion of powder is satisfactory, and it seems that it is impossible to predict with accuracy the composition of the residue, either gaseous or solid. The main point at issue is, however, demonstrated, and it is apparent that powder fired in the presence of coal produces a larger percentage of CO than when fired alone. This is, of course, to be expected, as the coal has very nearly the effect of the charcoal of the powder and the mixture of powder and coal behaves like a powder deficient in nitrate, and there is also some volatile combustible matter distilled from the coal.

It is not claimed that the conditions of the experiments duplicated in detail those attending the use of powder in coal-mining, but it is believed that they approach those conditions so nearly as to leave no room for question concerning the probability of the production of explosive gases from powder fired in coal. The powder, when fired in the mine, shatters the coal more or less and the hot gases pass over the surface of the coal and envelop the dust. This is especially the case when more powder is used than is necessary to perform the mechanical work of breaking the coal, for in that case the coal is broken into smaller pieces, having a larger total surface, and the opportunity for contact between the hot gases and the coal is more complete.

That the effect of coal in increasing the volume of gases produced is recognized by the miners, is shown by the custom, fortunately not common, of replacing a part of the powder by

coal-drillings. It has been stated by miners in the Kansas field that a charge of 6 lb. of powder and 1 lb. of coal-drillings would do the work of a charge of 7 lb. of powder. This seems to be the case.

It was noticed that, as the percentage of coal-dust was increased, the combustion became less rapid. The solid residue was also changed in character, becoming black and brittle, while that produced by powder alone was tough and usually yellow or red in color. This residue is easily soluble and is strongly alkaline in reaction. It is also seen that in the presence of large quantities of coal, the total quantity of gases produced is diminished.

III. Blown-Out Shot Experiments.

In order to determine, as far as possible by laboratory methods, the composition of the gases caused by a "blown-out" shot, an apparatus was employed consisting of a piece of 1-in. pipe, about 14 in. long, and closed at one end, used as a cannon. This was charged with powder or with a mixture of powder and coal, and fired into one end of a piece of 3-in. pipe, one end of which was closed with a cement plug having an opening for the cannon. The cannon could not be made to fit tight in the cement plug, and this plug also had an opening for the withdrawal of samples. When the powder was fired, the gases entered the collecting-pipe with such velocity as to cause a reduction of pressure at the firing end and air was drawn in.

The apparatus was slightly modified by substituting a clay plug for the cement, thus permitting the cannon to be luted tight for each explosion; by placing a weighted asbestos curtain, or flap-valve, over the open end of the collecting-pipe in order to reduce the velocity of the gases; and by inserting a small tube through the wall of the collecting-pipe for the withdrawal of samples. These changes allowed the collection of samples much less contaminated with air than those at first obtained.

A "blown-out" shot is one in which the explosive does not break the coal or rock as it is intended to do, but finding a path of less resistance, blows out the tamping, projecting the gases from the hole as from a gun. It was the purpose, in this set of experiments, to determine the nature of the gases evolved by the powder, burning alone under such conditions, mixed with coal-dust, and projecting its flame into an atmosphere charged with coal-dust.

The powder was so ignited as to burn backward from the mouth of the cannon, as would be the case with a blown-out shot, instead of projecting powder-grains out of the cannon.

The early analyses showed a large amount of oxygen in the gas. As this was undoubtedly due to the presence of air, the analyses were recalculated, ascribing all the oxygen to air, and giving the composition of the gases due to the burning of the powder and the distillation of the coal. This is not strictly accurate, since the powder alone had showed a little oxygen. However, the amount of oxygen found in the powder-gases was almost always very small, and as it was not constant, it seemed unwise to attempt to ascribe an arbitrary amount to the powder-gases and the rest to air. Therefore it is believed that, though these calculations do not represent the facts with absolute accuracy, they do represent them with so close an approximation that for the purposes of the experiments they may be considered accurate.

TABLE III.—*Composition of Gases Caused by a "Blown-Out" Shot.*

Test No. 1. 90 g. No. 7 Powder.

Gases.	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 \div H_2S$	22.6	41.1
C_2H_6	0.6	1.1
O_2	9.0	0.0
CO	3.4	6.3
H_2	0.8	1.5
CH_4	0.4	0.7
N_2	62.7	49.5
Total	99.5	100.2
Combustible	5.2	9.6

Test No. 2. 100 g. powder.[a] Sample practically air; discarded.

Gases.	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.

Test No. 3. 110 g. powder[a] with 11 g. pea-size coal from Monongah mine No 8.

Gases.	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 \div H_2S$	23.3	44.0
C_2H_6	0.3	0.6
O_2	9.6	0.0
CO	3.3	6.4
H_2	0.0	0.0
CH_4	0.4	0.8
N_2	63.1	48.4
Total	100.0	100.2
Combustible	4.0	7.8

Test No. 4. 100 g. powder[a] without coal.

Gases.	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 + H_2S$	9.8	36.1
C_2H_6	0.0	0.0
O_2	14.7	0.0
CO	0.8	2.9
H_2	0.7	2.6
CH_4	0.0	0.0
N_2	74.0	58.4
Total	100.0	100.0
Combustible	1.5	5.5

[a] Powder obtained from a local dealer.

Test No. 5. 100 g. powder[a] without coal.

Gases	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 + H_2S$	17.9	37.8
C_2H_6	0.0	0.0
O_2	11.1	0.0
CO	3.6	7.6
H_2	2.5	5.3
CH_4	2.5	5.3
N_2	62.5	43.9
Total	100.1	99.9
Combustible	8.6	18.2

At this point the apparatus was changed as previously stated.

Test No. 6. 100 g. powder[a] without coal.

Gases.	As Analyzed	Air Eliminated
	Per Cent.	Per Cent.
$CO_2 + H_2S$	32.3	47.3
C_2H_6	0.0	0.0
O_2	6.5	0.0
CO	4.0	6.0
H_2	0.0	0.0
CH_4	0.0	0.0
N_2	57.2	46.8
Total	100.0	100.1
Combustible	4.0	6.0

Test No. 7. 100 g powder[a] with about 40 g. coal-dust scattered along the large pipe. The violence of the combustion and the volume of smoke were much greater than in the preceding experiments.

Gases	As Analyzed	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 + H_2S$	37.8	43.1
C_2H_6	0.0	0.0
O_2	2.5	0.0
CO	11.4	12.9
H_2	1.9	2.2
CH_4	1.9	2.2
N_2	44.5	39.7
Total	100.0	100.1
Combustible	15.2	17.3

Test No. 8. 100 g. powder[a] with about 20 g. of coal-dust scattered as before.

Gases	As Analyzed	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 + H_2S$	35.9	38.0
C_2H_6	0.9	0.9
O_2	1.1	0.0
CO	12.9	13.6
H_2	1.9	2.0
CH_4	2.7	2.9
N_2	44.7	42.7
Total	100.1	100.1
Combustible	18.4	19.4

Test No. 9. 100 g. powder[a] mixed with 5 g. of coal-dust.

Gases.	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 + H_2S$	31.1	31.9
C_2H_6	0.5	0.5
O_2	0.5	0.0
CO	6.2	6.3
H_2	0.6	0.6
CH_4	1.2	1.2
N_2	59.9	59.4
Total	100.0	99.9
Combustible	8.5	8.6

Test No. 10. 100 g. powder[a] mixed with 20 g. of coal-dust.

Gases.	As Analyzed.	Air Eliminated.
	Per Cent.	Per Cent.
$CO_2 + H_2S$	39.9	41.1
C_2H_6	0.6	0.6
O_2	0.6	0.0
CO	7.3	7.7
H_2	0.0	0.0
CH_4	3.1	3.3
N_2	48.5	47.4
Total	100.0	100.1
Combustible	11.0	11.6

[a] Powder obtained from a local dealer.

The results given in Table III. show a greater proportion of combustibles from those experiments in which the coal was scattered along the pipe in front of the powder. If not actually combustible, the gases were, in some cases, very near the limit of combustibility, and it would seem that a combustible gas-mixture might easily be produced.

Some of the gas-mixtures are so near the point of combustibility that they would be made combustible by a small admixture of methane or coal-dust; for it has been demonstrated in another set of experiments that the presence of combustible gases in the air, even in small amounts and well below the limit of combustibility, will render explosive an atmosphere containing coal-dust which is present in too small amounts to be combustible alone. It is evident, then, that the presence of such gases in the mine-air must be considered a source of danger. This is especially the case when the gases are produced by black powder, as they might easily be ignited by the flame from a subsequent shot; but it would also seem to be the case if the gases are produced by some other means. Some of the "safety" powders on the market produce large quantities of carbon monoxide, and it seems that the presence of this gas in the mine atmosphere may not be unattended by danger even though the flame of the powder itself will not ignite the gas. This is a point which has apparently received little attention and it is to be hoped that notice may not be drawn to it by the occurrence of some serious disaster.

IV. Effect of Pressure at Time of Explosion.

As one of the points demonstrated by Noble and Abel in their long series of experiments was that the percentage of CO increased as the space occupied by the powder in the combustion-chamber decreased, that is, as the temperature and pressure decreased, it seemed that shots in which the charge of powder was unnecessarily large might give rise to gases in which the amount of CO would be so great as to be a source of danger, because in such shots the coal would be broken down before all of the powder had been consumed and a portion of it would burn at reduced pressure. In order to compare the behavior of American blasting-powders with the ordnance-powder which was used by these English experimenters, a series of explosions

was made in which different amounts of powder were ignited in the same chamber. This was the method employed by Noble and Abel. When confined, the charges were ignited in the bomb; when not confined, in the retort. The experiments require no further description. For the first set, bomb No. 1 was used, the volume of its chamber being about 22 cubic centimeters.

TABLE IV.—*Results of Exploding Different Quantities of Powder in the Same Chamber.*

Charge.	Percentage of Volume of Chamber Occupied by Powder.	CO_2 + H_2S.	C_2H_6.	O_2.	CO.	CH_4	H_2	N_2.	Total.	Combustible.
			Powder No. 4.							
	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
10 g., ...	45.5	57.0	0.1	0.6	6.0	1.1	0.0	34.8	99 6	7 2
2.5 g., ...	11.3	61.6	0.2	0.1	4.0	0.3	0.0	34.3	100.5	4.7
Retort at atmospheric pressure,	0.0	54.5	0.0	0.0	3.3	0 0	0 0	42 4	100.2	3 3
			Powder No. 5.							
10 g., ...	45.5	58.6	0 0	0.4	7.2	0.4	0.0	33 0	99 6	7.6
5 g., ...	22.7	61.2	0.6	0 6	4.2	0.0	0.0	33.9	100.5	4.8
2.5 g., ...	11.3	62.2	0.0	1.0	1.5	0.0	0.0	35.0	99.7	1.5
Retort at atmospheric pressure,	0.0	45.5	0.4	1.6	1.9	0.2	0.0	51.1	100.7	2.5
			Powder No. 6.							
10 g., ...	45.5	58.0	0.0	0.3	7.2	2.0	0.0	32.5	100.0	9.2
5 g.	22.7	58.9	0.4	0.2	6.7	0.0	0.0	33 4	99.6	7.1
2.5 g., ...	11.3	61.7	0.2	0.1	4.0	0.3	0.0	34.3	100.6	4.5
Retort at atmospheric pressure,	0.0	43.2	0.2	1.6	3 0	0.2	0.0	51.7	99.9	3.4
			Powder No. 7.							
10 g., ...	45.5	58.7	0.0	0.0	7.2	0.3	0.4	33.4	100.0	7.9
Retort at atmospheric pressure,	0.0	41.1	1.1	0.0	6.3	0.7	1.5	49.5	100.2	9.6

These experiments show a decrease in the percentage of CO accompanying the decrease of gas-pressure, with the exception of powder No. 5 exploded in the retort, and even in this case the percentage of CO is very low. It will be noticed that the percentage of nitrogen in the gases obtained from the retort is very high. Apparently the conditions are very different from those attending explosion in the bomb.

Experiments were made with powder purchased from a local dealer. Bomb No. 2 was used, the volume of the combustion-chamber being about 50 cc. The results are given in Table V.

TABLE V.—*Results of Exploding Different Quantities of Powder in the Same Chamber.*

Charge.	Percentage of Volume of Chamber Occupied by Powder.	CO_2 + H_2S.	C_2H_6.	O_2.	CO.	CH_4.	H_2.	N_2.	Total.	Combustible.	
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	cc.
16 g., ...	32.0	59.7	0.4	0.1	3.7	0.7	0.3	35.2	100.0	5.1	4,200
8 g., ...	16.0	61.6	0.3	0.4	2 8	0 8	0.0	34.1	100.0	3.9	2,300
4 g., ...	8.0	60.8	0.2	0.3	1.2	0.3	0.1	37.1	100.0	1.8	800
Retort at atmospheric pressure,	0.0	43.0	0.0	0.0	5.7	1 8	1.8	47.8	100.1	9.3	

The experiments in Table V., like the preceding ones, show a decrease in the percentage of CO accompanying decreased gas-pressure when the gases are confined. It is seen that the percentage of CO is increased when the powder is burned in the retort, and that the percentage of nitrogen also increases. It will also be noticed that the powder when confined gives a very small percentage of CO. This powder is the same as that used for the second series of experiments with powder in the presence of coal-dust, in which a marked increase of combustible gases was seen to be produced.

These experiments give results opposite to those recorded by Noble and Abel, in which the percentage of CO increased as the pressure of gases decreased. Apparently the difference arises from the difference in the powders used, the American powders containing a larger percentage of nitrate than the English mining-powder, as has been previously stated. In the absence of a theory which will satisfactorily account for all of the phenomena of the combustion of powder, no attempt is made to explain this difference in results. It is to be hoped that such a theory will appear, allowing a prediction of the combustion-products formed under differing conditions.

V. EFFECT OF MORE OR LESS THOROUGH MIXING OF THE INGREDIENTS.

One step in the manufacture of black powder is the grinding together, or incorporation, of the ingredients, which is performed by heavy rollers running on a circular track. The time allowed for the incorporation of a charge varies somewhat in different mills, and possibly in any mill it may not always be the same. It was thought that the thoroughness of incorpo-

ration might have an effect upon the decomposition-products. In order to test this point samples were obtained from two mills, in each of which the powder is customarily ground for 2 hr. As the powder is ground wet and dries to some extent during grinding, samples taken at different points in the process contain different amounts of water. It was necessary to make the moisture-content uniform in order to obtain comparable results, and the powders were dried in an electric oven at a temperature of 105° C.

In order to ascertain whether an appreciable amount of sulphur might be volatilized at this temperature a test was made, giving the following results:

	Per Cent.
Sulphur, dried 4 hr. .... at	100° C., loss = 0.3
Sulphur, dried 8 hr. longer at	90° C., loss = 0.1
Sulphur, dried 6 hr. longer at	105° C., loss = 0.6
Sulphur, dried 4 hr. longer at	105° C., loss = 0.2

This shows a comparatively large loss during the first 4 hr. at 100° C. Undoubtedly this loss is mostly, if not entirely, due to the evaporation of water. A larger loss occurs during the earlier stages of drying at 105°. The fact that the loss at this temperature is not uniform during the time of drying, as shown by the decrease in weight of only 0.2 per cent. during 4 hr. longer exposure to this temperature, suggests that the loss is probably due to the further evaporation of water which was not driven off at the lower temperature. However this may be, it is evident that the question of the volatilization of sulphur may be neglected in the consideration of the point at issue.

The first tests were made on samples taken as follows:

No. 1, incorporated, ...	...... 15 min.
No. 2, incorporated, ...	...... 45 min.
No. 3, incorporated, ...	1 hr. 15 min.
No. 4, incorporated, ...	1 hr. 45 min.

Table VI. gives a summary of results obtained by making from six to eight experiments with each powder, several experiments being omitted for the sake of brevity.

TABLE VI.—*Summary of Results—Thoroughness of Mixing.*

No. of Sample.	1.	2.	3.	4.	1.	2.	3.	4.
Temp. of drying, deg. C., . . .	105°	105°	105°	105°	......	......	......	......
Time dried, hr.,	4	4	4	4	......	......	......	......
Charge, g., . .	8	8	8	8	8	8	8	8
Volume of gas, cc, ...	2,700	......	2,300	2,450	1,860	2,270	2,000	2,100
	Per Cent.	Per Cent.	Per Cent.	Per Cent	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Moisture, . . .	0	0	0	0	4.1	3.4	.....	.....
$CO_2 + H_2S$ . .	56.2	56.2	52.8	54.7	56.1	56.5	56.4	55.0
C_2H_6	0.2	0.1	0.2	0.2	0.1	0.1	0.1	0.5
O_2	0.1	0.2	0.2	0.5	0.2	0.1	0.2	0.0
CO	9.1	8.9	10.2	9.6	7.2	7.4	8.1	9.7
CH_4	0.2	0.2	0.2	0.2	0.2	0.1	0.1	0.2
H_2	1.1	1.2	1.6	1.3	1.7	1.7	1.2	1.7
N_2	33.2	33.7	34.8	33.5	33.8	34.4	33.9	33.1
Total, ...	100.1	100.5	100.0	100.0	99.3	100.3	100.0	100.2
Combustible, per cent., . . .	10.6	10.4	12.2	11.3	9.2	9.3	9.5	12.1

No apparent relation is revealed between the thoroughness of incorporation and the composition of the gases, but it is noticeable that in most cases the percentage of combustible gases is lower in the samples which have not been dried. The experiments the results of which are not detailed are fairly uniform in this respect. It is also seen that the difference is in the percentage of CO.

In further pursuance of the investigation of this point tests were made with three powders from another mill. No. 1 is a powder of medium grain, No. 2 of large grain, and No. 3 of fine grain. There were five samples of each, as follows:

A, mixed, ...	30 min.
B, mixed, ...	1 hr.
C, mixed, ...	1 hr. 30 min.
D, mixed, ...	2 hr.
E, mixed, ...	finished powder.

As before, not all of the experiments are recorded.

TABLE VII.—*Results of Exploding Samples Taken at Different Stages of Mixing.*

Powder.	1 A.	1 B.	1 C.	1 D.	1 E.	1 A.	1 B.	1 C.	1 D.	1 E.
Temp. of drying, deg. C. . . .	105°	105°	105°	105°	105°	....	.. .	....	... .	.. . .
Time dried, hours,	4	4	1	1	4	.	. . .	. . .	. .	. . .
Charge, g., . .	8	8	8	8	8	8	8	8	8	8
Volume of gas, cc.,	1,700	2,500	2,100	2,750	2,550	2,500	2,100	. . .	2,550	2,750
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Moisture, . . .	0	0	0	0	0	4.7	1.1	2.9	1.1	.
$CO_2 + H_2S$ . .	52.7	55.9	51.0	55.0	55.2	58.1	56.8	56.9	56.5	51.9
C_2H_6	0.1	0.3	0.0	0.3	0.5	0.3	0.1	0.0	0.1	0.2
O_2	0.6	0.2	0.1	0.1	0.1	0.3	0.1	0.1	0.3	0.1
CO	7.3	6.6	7.9	8.2	8.0	2.2	5.8	6.1	7.0	7.5
CH_4	0.1	0.2	0.1	0.1	0.3	0.0	0.2	0.1	0.0	0.1
H_2	1.1	1.2	1.2	0.7	0.8	1.0	1.8	1.3	0.5	0.7
N_2	37.5	35.1	36.6	35.6	35.6	35.9	31.9	35.5	35.5	36.5
Total, ...	100.0	99.8	99.9	100.0	100.5	100.8	100.0	100.0	99.9	100.0
Combustible, per cent., ...	9.2	8.3	9.2	9.3	9.6	6.5	8.2	7.5	7.6	8.5

Powder.	2 A.	2 B.	2 C.	2 D	2 E.	2 D.	2 E.	3 A.	3 B.	3 C.	3 D.	3 E.	3 D.
Temp. of drying, deg. C. . . .	105°	105°	105°	105°	105°	....	. . .	105°	105°	105°	105°	105°	... .
Time dried, hours,	4	1	4	1	1	.	. ..	1	4	1	1	1	. . .
Charge, g., . .	8	8	8	8	8	8	8	8	8	8	8	8	8
Volume of gas, cc.,	2,750	.. .	2,750	2,700	2,550	2,600	2,600	2,550	2,800	2,800	2,600	.	2,750
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Moisture, . . .	0	0	0	0	0	1.6	. .	0	0	0	0	0	
$CO_2 + H_2S$ . .	56.5	51.7	51.1	55.0	56.1	56.2	58.1	58.1	59.3	57.1	56.2	58.2	60.2
C_2H_6	0.1	0.1	0.1	0.3	0.3	0.5	0.2	0.2	0.2	0.3	0.1	0.3	0.1
O_2	0.2	0.2	0.2	0.0	0.1	0.2	0.3	0.2	0.3	0.1	0.2	0.1	0.2
CO	8.1	9.1	9.8	8.9	8.2	7.9	6.5	5.3	5.6	7.0	8.2	6.5	1.3
CH_4	0.2	0.2	0.2	0.0	0.2	0.0	0.1	0.2	0.2	0.1	0.1	0.2	0.1
H_2	0.8	0.1	1.1	0.8	0.8	0.9	0.5	0.5	0.7	0.1	1.0	0.3	0.6
N_2	33.9	35.0	31.3	31.9	31.0	31.1	31.0	35.6	33.7	31.9	33.9	31.1	31.0
Total, ...	100.1	100.0	100.1	99.9	100.0	100.1	100.0	100.2	100.0	99.9	100.0	100.0	100.1
Combustible, per cent., ...	9.5	10.1	11.2	10.0	9.5	9.3	7.3	6.0	6.7	7.8	9.7	7.1	5.6

It is evident that the experiments in Table VII. show no regular effect of more or less thorough incorporation. It is not to be expected that a small grab-sample, weighing perhaps 0.5 lb., will be truly representative of the composition of a mass of several hundred pounds until the incorporation is complete. The fact that these samples show no important differences in the composition of the gases given off indicates that the incorporation is more thorough early in the process than has been supposed. The results also indicate that no serious trouble is likely to arise from any slight difference in the length of time of incorporation.

VI. Effect of the Presence of Moisture on the Composition of the Gases.

As it has been said that the gases from a powder which has absorbed moisture contain higher percentages of combustibles than those from dry powders, experiments were performed to test the statement. In some cases the powder was exploded without either wetting or drying, in others it was dried, and in others dried and then allowed to absorb moisture. The last procedure was adopted in order to meet the argument that the process of drying had caused some change in the composition of the powder other than the removal of moisture. The previous tests of the volatilization of sulphur would seem to have made this precaution unnecessary, but it was taken with the idea of making the demonstration plain. The powders which were moistened were allowed to absorb moisture from air saturated by steam.

The first test was made on a museum specimen which had slowly absorbed moisture in the course of five or six years.

TABLE VIII.—*Effect of Moisture on Composition of Exploded Gases.*

Condition	Neither Wet nor Dried.	Wet	Wet
Moisture, per cent.,	2.5	3.9	13.8
Charge, g.,	8	8	8
Volume of gas, cc.,	2,400	2,650	2,100
	Per Cent	Per Cent.	Per Cent
$CO_2 + H_2S$	56.3	56.8	57.9
C_2H_6	0.3	0.5	0.3
O_2	0.2	0.2	0.3
CO	8.0	7.1	3.9
CH_4	0.1	0.5	0.3
H_2	1.0	1.6	1.3
N_2	34.1	33.9	35.6
Total,	100.0	100.6	99.6
Combustible, per cent.,	9.4	9.7	6.9

The sample containing the largest percentage of moisture gave a solid residue which was moist, black, and soft, while the residue from powders containing small amounts of moisture was always very compact.

Table IX. gives results of tests with the powder purchased from a local dealer.

TABLE IX.—*Effect of Moisture on Composition of Exploded Gases, Using Local Powder.*

Condition.			Dry.	Wet.	Wet and Dried.	Wet and Dried.	Wet and Dried.			
Moisture, per cent.,	1.4	1.4	0.1	12.6	8.0	1.8	1.2	1.3	1.4	1.4
Temp of drying, deg. C.,	.....	......	105°	......	room	room	105°	....	.....	......
Time of drying, hr,	.....	....	4	......	5	12	4	....	.....	......
Volume of gas, cc.,	2,450	2,250	2,200	2,350	2,400	2,100	2,100	2,400	2,200	......
Charge, g	8	8	8	8	8	8	8	8	8	8
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
$CO_2 - H_2S$	60.7	61.7	60.4	62.8	63.4	62.2	62.6	62.4	62.2	62.6
C_2H_6	0.9	0.6	0.3	0.2	0.4	0.3	0.4	0.7	0.5	0.3
O_2	0.2	0.2	0.2	0.3	0.3	0.1	0.2	0.4	0.4	0.0
CO	3.6	2.3	3.9	1.0	0.5	1.6	1.7	1.9	1.9	2.4
CH_4	0.0	0.1	0.1	0.1	0.1	0.0	0.1	0.1	0.0	0.2
H_2	0.0	0.0	0.1	0.4	0.1	0.0	0.1	0.2	0.4	0.1
N_2	34.6	35.2	34.9	34.5	35.2	35.8	35.1	34.3	34.6	34.4
Total,	100.0	100.1	99.9	99.3	100.0	100.0	100.2	100.0	100.0	100.0
Combustible, per cent.,	4.5	3.0	4.6	1.7	1.1	1.9	2.3	2.9	2.8	3.0

Because this powder gives a very small percentage of combus-
ıle gases, Table X. is added to show the results of tests with
e powders used for testing the effect of thoroughness of in-
rporation.

.BLE X.—*Effect of Moisture on Composition of Exploded Gases, With Other Powders.*

'owder mple No.	1 A.	1 A.	1 B.	1 B.	1 C.	1 C.	1 D.	1 D.	2 D.	2 D.	3 D.	3 D.
dition, .	. .	Dried	......	Dried	..	Dried	..	Dried	.....	Dried	.....	Dried
sture, per nt., . .	4.7	0	4.1	0	2.9	0	1 4	0	1 6	0	.	0
ıp. of dry- g, deg C,	...	105°	. .	105°	. .	105°	.....	105°		105°	....	105°
ıe of dry- g hr,	.	4	.	4	...	4	....	4	....	4	. .	4
iume of s, cc,	.....	1,700	2,400	2,500	.....	2,100	2,550	2,750	2,500	2,700	2,750	2,600
rge, g, . .	8	8	8	8	8	8	8	8	8	8	8	8
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
+ H_2S .	58.5	52.7	56.8	55.9	56.9	54.0	56.5	55.0	56.2	55.0	60 2	56.2
. . . .	0.2	0.4	0.4	0.3	0.0	0.0	0.1	0.3	0.5	0.3	0.4	0.4
. . .	0.3	0.6	0.1	0.2	0.1	0.1	0.3	0.1	0.2	0.0	0.2	0 2
. . .	4.4	7.3	5.8	6.6	6.1	7.9	7.0	8.2	7.9	8.9	4.3	8.2
. . . .	0.1	0.1	0.2	0.2	0 1	0.1	0 0	0.1	0.0	0.0	0 3	0 1
. . .	1.5	1.4	1.8	1.2	1.3	1.2	0.5	0.7	0.9	0.8	0.6	1.0
. . .	34 7	37.5	34.9	35.4	35.5	36.6	35.5	35.6	34.4	34.9	34.0	33.9
ıl, . .	99.7	100.0	100.0	99.8	100.0	99.9	99.9	100.0	100.1	99.9	100 0	100.0
ıbustible, r cent, . .	6.2	9.2	8.2	8.3	7.5	9.2	7.6	9.3	9.3	10.0	5 6	5.7

It is very evident from the tests in Tables VIII., IX., and X., ıt the presence of even a small amount of moisture results a diminution of the percentage of combustibles in the gases. ıis result, which is contrary to what has been generally be- ved and to what was expected, is of course to be desired. is not apparent that the presence of a small percentage of ɔisture produces any regular effect upon the volume of the ses. It must, however, reduce the temperature of the gases the moment of explosion, and therefore reduce the force of e explosion. It should therefore somewhat reduce the lia- ity to ignite combustible gases or dust, but this result is tter attained by some of the safety powders which have the ded advantage of short flame and short duration of high nperature.

VII. Conclusion.

By way of conclusion it may be said that American black powder, fired alone, does not produce a combustible gas. When fired in the presence of coal, however, it may produce such a gas. The fact that powder-gases not infrequently become ignited and burn in the mine is evidence that the conditions for the production of a combustible gas are met in the actual use of the powder.

Probably the gas produced in the mine is not always combustible. It is evident from the results of analyses given before that a portion of the combustibles is produced by distillation of the coal. This being true, it appears that a coal containing small amounts of volatile combustible matter should yield a gas-mixture comparatively low in combustibles, and the danger from the use of black powder should be least with such coal. On the other hand, with coal rich in volatile combustibles, the gas produced is likely to be combustible, and the danger to be the greatest.

Probably the composition of the gases is influenced also by the amount of powder used. If the hole is properly charged and tamped, the coal will be broken down with little shattering and with the production of little dust. In this case only the outsides of the lumps will be heated, because the hot gases will be present in quantities too small to heat the lumps thoroughly, and only a small part of the volatile matter will be driven off. On the other hand, when too much powder is used, as is very frequently the case, the coal is more thoroughly shattered, and the hot gases, present in large quantities, act on pieces of coal so small as to be more thoroughly heated and large quantities of volatile matter will be driven off. And even if the interiors of the lumps are not heated, the increased surface exposed by the more shattered coal affords opportunity for the distillation of more gas.

The fact that combustible gases and coal-dust may replace each other in a combustible mixture with air, increases the danger from the use of black powder or any other explosive yielding gas-mixtures containing combustible constituents. The danger is especially great in cases where more than one shot is fired in a room. Black powder is almost always ignited by a fire-fuse and other explosives are commonly so fired. The

shots do not explode simultaneously, and when the last shot is fired there is already present in the room the gas from one or more previous shots, and this gas may be combustible and may be mixed with air in the proper proportion to be explosive. In addition to this, the dust in the room has been disturbed by the blast of air from the previous shots and is in suspension. It is evident that the conditions may be such as to yield a gas- or dust-explosion within the room if the mixture can be ignited. The explosion of the last shot may furnish the igniting flame. Black powder produces a large flame of long duration and high temperature. In most cases the flame is not projected into the room, but the hot gases are cooled by passing over the surface of the broken coal. But if the last shot should blow out, the conditions for an explosion may be reached. This explosion is in most cases confined to the room in which it originates, but in case flame should pass from the neck of the room to a dusty entry the explosion may be propagated and assume serious proportions.

Certain safety-powders produce gases containing so much carbon monoxide, methane, and hydrogen as to be combustible. Their safety lies in the fact that they give a small flame of short duration, incapable of igniting a combustible gas. As long, then, as there is no other source of ignition, no explosion can result. But it seems possible that the gases might be ignited in some other way.

An increase of danger results from the fact that in some cases many shots are fired at nearly the same time. This is the case in Kansas, where all shots are fired by shot-firers, who do their work after all others have left the mine. In Kansas there is supposed to be one shot-firer for every 40 miners or any fraction thereof. The shot-firers work as rapidly as possible. Under these conditions, not only a single room, but a large portion of a mine, is filled with a mixture of dust, powder, gases, and air, which may be explosive. This is undoubtedly the reason for many explosions. As a rule, it is impossible to ascertain the exact conditions, because in case of explosion the shot-firer is generally killed.

It is evident also that the presence of a large amount of carbon monoxide in the air is a source of danger because of the poisonous nature of the gas. In case of promiscuous firing

the mine-air may constantly contain a small percentage of this gas, and its effect on the health of the miners cannot be other than bad. When shot-firers are employed they alone suffer. In most cases they are able to keep out of the gas, but suffocation is by no means infrequent.

The present tendency is undoubtedly towards the use of more powder than is desirable. It is easier for the miner to shoot the coal than it is to mine it with a pick. There are three undesirable effects: (1) the roof is weakened and the number of accidents from fall of roof is increased; (2) the coal is shattered and its value decreased; (3) large quantities of powder-gas are produced. The first two effects are greatly increased by the use of dynamite, a practice which is indefensible, but which it seems impossible entirely to prevent.

In case the miners are paid for run-of-mine coal it is impossible to restrict greatly the use of explosives. When payment is made for lump-coal the miners are more careful, because the use of much powder decreases the proportion of lump-coal and therefore decreases the earnings of the miner. But even in this case the use of explosives may result in the breaking-down of so large a quantity of coal as to compensate for the decrease in the proportion of lump, giving the miner as large earnings as he would receive if he produced less coal with a larger proportion of lump, and making the work of getting it easier. In Kansas it is required by law that payment be made on a run-of-mine basis, and the result is the production of a large amount of slack. It seems to me that the arrangement obtaining in Michigan is more satisfactory. In this State the operator pays for run-of-mine or for screened coal at his own option and the miner does not know on which basis he will be paid at any time. The miners have a representative at the scales, and the rate for screened coal is higher, so the arrangement seems fair to both operator and miner.

In Kansas the tendency towards the excessive use of powder is further increased by the fact that the powder-jack has been abolished, and the operators are required to furnish powder in 12.5-lb. cans. As no powder is allowed to remain in the mines and the miners will not carry it back to the surface, each man is practically certain to use a can each day, whether he needs it or not.

It appears that the objections to the use of black powder are not founded upon any question of the good or bad quality of the powder, for though many samples were examined, all were found to be good. The objectionable features are inherent in the nature of the explosive, but may be aggravated by the method of its use. But though it is seen that the gases produced may, under some circumstances, be combustible, it must be remembered that when the proper charge is used the danger is a minimum, and this explosive, being slow-burning, shatters the coal less than the more violent explosives. It is believed that black powder may continue to have a considerable use in mines free from gas, and in which the dust is sufficiently damp to prevent suspension in the air.

Combustion in Cement-Burning.

BY BYRON E. ELDRED, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910.)

GENERALLY speaking, the practical study of combustion has been made mainly from the stand-point of the steam engineer. This narrow view-point has left open a large field for scientific research on the application of combustion in other arts, where the conditions of heat-utilization are widely different from those presented in the steam-generator. The great economic importance of the boiler and the ease of measuring its performance, together with the comparative uniformity of conditions in the steam-generator, is, probably, quite largely responsible for the practical neglect of the study of the use of fuels in other arts save only on the basis of their thermal value.

Yet these other arts show quite plainly that a thermal-value basis alone is not sufficient. For example, in burning lime, under ordinary conditions of firing, a cord of wood will do as much work as a ton of coal, though the latter affords more than twice the number of heat-units. Similar results are evidenced in burning brick, in the roasting of certain ores, in reheating billets, and in many other arts where heat is to be applied over extended areas to fragmental masses of more or less refractory materials. In these arts much is to be gained by a

careful study of the application of the heating-agent in the form of the long flame, and in the production of the heat in contact with, or in direct proximity to, the object to be heated. Of necessity, in boiler-practice, combustion cannot be maintained in actual contact with, or in close proximity to, the object to be heated.

Speaking in a general way, heat is imparted from burning bodies to things to be heated in two ways: directly, by radiation, and indirectly, by transfer of the sensible heat of hot gases by convection or contact; by conduction from the hot gases. Both ways are utilized in boiler-practice—directly, by the radiation from the flame and from furnace-walls, and indirectly, by the heating-effect of currents of hot-flame gases. But because the thing to be heated, the water or its metal container, is always at a temperature which will extinguish flame, the heat is always generated elsewhere and conveyed to the boiler; it cannot be produced at the point where it is wanted. Again, the very wide and essential difference between the heating of water and the work performed by heat in other arts lies in the property of the liquid to distribute the heat evenly throughout its mass.

Different conditions prevail in other arts; but these conditions have not been well studied, and our practice is crude. As regards radiation, the old axiom that the power of a body to intercept heat is inversely proportional to the square of its distance from the source of the heat, is still worth consideration. The application of pre-generated heat to extended areas of fragmental, more or less refractory materials, in the same way as in the boiler-furnace, is wasteful and unsatisfactory. This is the reason for the preference of long-flame fuels for certain work, and the reason why gas-producers are run with shallow fuel-beds for soaking-pits and billet-reheating furnaces; the reason why lean gas is wastefully produced at a much greater expense than the richer gas from the more economically operated producers that supply the open-hearth furnaces. The lean gas gives a different type of flame; a slower, longer flame suitable for the comparatively low, slow heating of solid masses over extended areas.

This cited case of wasteful heat-generation to secure special application is a fair support for my statement that the study of

the phenomena of applied combustion has been neglected. The fact has never been pointed out that the reason for this wasteful low-fire producer-operation is to secure later the flame-results of a lean gas carrying a large percentage of the neutral diluent gases, nitrogen and carbon dioxide, though, as respects the latter, it is perhaps hardly to be regarded as a purely neutral diluent. The gas-producer operator knows only this indirect way of securing a desirable end, while to any man of ordinary scientific knowledge, the mere suggestion of the end to be attained should suggest the better way of an economical manufacture of the producer-gas and the later use of waste gases in admixture with air to produce the desired type of flame: a way which is susceptible of regulation and control.

In my studies of long-flame combustion, I have noted remarkably wide differences, and great advantages, in the application of the heating-agent to generate heat in contact with the work, as contrasted with the prior methods of applying pre-generated heat. The practical workings of this idea were investigated by a committee of the Franklin Institute about five years ago and reported on favorably; therefore, I will not go into further detail, and only offer the foregoing as a preface to the statement that I consider cement-burning as conducted in present-day accepted American practice a good demonstration of the crudeness of our application of the principles of combustion; and, to go a step further, of our lack of observation of the phenomena of combustion.

In the present practice of burning cement, the clinker is made by slowly passing a stream of finely-powdered cement-materials through a rotary kiln of from 5 to 8 ft. internal diameter and from 60 to 150 ft. long. In the entrance of the kiln burns an aërially-supported flame-plume, generally axially directed, produced by powdered coal injected by air. In passing through the kiln, the materials, which are an artificial or a natural mixture of clay and calcium carbonate, are first dehydrated by the heat of the flame-gases. Then the carbonate is causticized by dissociation, or calcined, the carbon dioxide being driven off by heat, and finally the clay and lime unite to form the clinker.

Each of these operations, dehydration, calcining, and clinkering, has different thermal requirements, and yet in the ordi-

nary practice all three are performed by one flame: a flame particularly adapted to the needs of the clinkering-operation and to that only. Clinkering requires a high temperature—a temperature sufficient to sinter the clay component and cause it to unite with the lime; but no great absorption of heat occurs; the action is exothermic. And as the calcines pass slowly under the blazing, heat-radiating coal-flame, they rapidly acquire the necessary temperature, and clinker. For the calcining, the waste heat of the flame-gases is considered all that is necessary; yet the marked economy of the slow-burning, voluminous flame for calcining is well known in other arts.

The net result is that the ordinary cement-kiln is a good clinkerer, a highly inefficient calciner (the calcines often come into the clinkering-zone with a material percentage of their carbon dioxide remaining), and a very wasteful thermal device.

So far as combustion is concerned, the kiln is a horizontal flue, and, as in every other horizontal flue, whatever gas-stratification can occur will occur. Much does occur. The flame is purposely spaced away from the work in the clinkering-zone, and the gases from it rise into the arch of the kiln and never get into effectual contact with the work in the calcining-zone. The heating in the calcining-zone is largely by radiation from the upper walls, and by direct conduction as the upper wall becomes the lower and moves under the onflowing stream of material. There is practically no direct heating by the flame-gases. These gases flow along the upper part of the kiln, and under them flows an entrained current of relatively colder air, partly brought in by the stack-suction and partly by the injector-action of the flame and coal-feeder.

Not only is the heating of the calcining mass inefficient, but the calcination is furthermore performed under the worst possible conditions. Calcination is a heat-absorbing reaction, and the evolved carbon dioxide is, therefore, relatively cold and dense, while it is naturally a gas of high specific gravity. It, therefore, tends to remain in the bottom of the kiln, flowing along as a separate stratum bathing the material, and very little admixed with the gases and air above. In other words, the carbonate is being calcined in an atmosphere of carbon dioxide; or, at least, an atmosphere which is abnormally rich

in carbon dioxide. Now calcination is one of the reactions which, like evaporation, depends very much upon the ambient atmosphere. Other things being equal, and within certain ranges of temperature, the heat required to dissociate calcium carbonate is inversely as the percentage of carbon dioxide in the atmosphere. That is one of the reasons why the rotary kiln is so poor a calciner as compared with the vertical, separately-fired lime-kiln. In the latter, the pieces of dissociating limestone are always bathed in a constantly-changing atmosphere, which can never contain more than 21 volume per cent. of carbon dioxide (the theoretical maximum for products of combustion), and which, as a matter of fact, never runs anywhere near as high as that amount.

In the vertical kiln, calcination and heat-utilization are very much better than in the rotary kiln, which is essentially a device adapted for the use of cheap fuel and expensive labor. And calcination, though not generally so recognized, is one of the most important functions of the cement-kiln. In the ordinary rotary kiln, the material comes into the vicinity of the clinkering-flame but partly calcined, and there is a sudden evolution of carbon dioxide at this point, disturbing the normal and orderly sequence of the operations and leading to the formation of clinker-rings and nodules.

As a labor-saving device the American coal-fired rotary cement-kiln was a great advance in the art over anything ever known before; but from the view-point of the heat engineer it is a grossly wasteful and inefficient apparatus. Though, theoretically, 25 lb. of coal should suffice for making a barrel of cement, the actual consumption is from 90 to 110 lb. Many attempts have been made to lessen this waste of fuel; but they have not embodied a scientific application of fundamental principles and, hence, have not been successful. Regenerators, recuperators, and the like cannot be employed with the present single kiln, for the reason, among others, that the dust evolved in calcining is carried forward by the violent rush of gases through the kiln and would soon choke up such heat-recovering devices, and also because of the necessity for unimpeded draft under the conditions of high localized heat-generation with the burning of fuel, taking into account the large air-excess requirements and the increase in volume of gases due to calcination-products.

Succinctly, in the ordinarly kiln two wholly-unlike operations, clinkering and calcining, are performed by one flame; and the results, so far as calcination is concerned, are not good, while the heat-waste is enormous. It is obviously better to perform the two stages as two stages in two kilns, and use for each combustion-methods best adapted for that particular stage. In doing this, moreover, the possibility of using heat-recovering devices is afforded, since, as it so happens, the dusting is mostly obviated by slow-traveling flames in both steps, while the high-temperature gases from the clinkering-operation can be used in regenerating the fuel for clinkering, and the lower-temperature gases from calcining can be used in regenerating for that operation, if desired, although this is less practicable. With the possibility of using regenerators comes the opportunity to use producer-gas, which, normally, gives rather too low a temperature for clinkering, in place of the more expensive powdered coal.

With this severing of the calcining from dependence upon the clinkering comes the practicability of a scientific utilization of combustion in the calcining-operation. The latter becomes a simple, lime-burning step, differing only in that the material, being finely powdered, cannot be handled in a vertical kiln and thus brought into actual contact with the flame and hot gases from ordinary firing-means. Something more than the ordinary type of boiler-furnace firing-means is necessary with fine material traveling down the rotary kiln. There must be a different type of flame: a flame filling the cylinder, and coming into actual contact with the material to be heated, and there burning, in place of traveling along its top. It is, of course, possible to produce combustion in contact with material which, unlike water-cooled boiler-tubes, can be raised over a red heat; and in so doing, the law of the square of the distance is no longer so important. As a matter of fact, combustion is actually much more vigorous in contact with suitable refractory materials, such as lime, than elsewhere; a fact which will be apparent to any one who has seen the flames licking the furnace-walls. Recent scientific work has shown that this "wall action" much heightens the velocity of combustion and, concomitantly, the number of heat-units which can be evolved in a time-unit.

There are long-flame fuels; but few of them can be economically used in calcining cement-materials, and fewer still will afford a flame capable of filling the barrel of the kiln. The flame of ordinarily good producer-gas is too short and scanty; and if it be burnt with regenerated air, as heat-economy dictates, it becomes still shorter. But the fat, long flame can be secured by the use of expedients which, looked at in one way, may be regarded as a development of the crude scheme of running the producer with an uneconomical "low fire." It is merely necessary to dilute the gas with a neutral gas, or, which is simpler, to dilute the air used for combustion. Or a triple mixture of air, diluent gas, and combustible gas may be made. Let us consider this last as the simplest case. If a mixture of combustible gas and air in theoretical proportions be made, ignition, of course, results in an explosion; that is, in a practically instantaneous combustion. When one portion of the gases is brought above a red heat, combination takes place and heat enough to carry on the reaction is evolved simultaneously. Now let us progressively dilute the mixture with an inert gas, say nitrogen. As the dilution goes on, the mixture becomes less and less explosive, till finally a point is reached where the ignition will no longer propagate itself through the mass at ordinary temperature. A spark or a flame brings the mixture in its immediate vicinity to a red heat and induces a local combination; but the ignition does not propagate through the gas mass. It will not burn. Now let us raise the temperature of the gas mass. Obviously it will again burn, and in burning the gas will evolve heat. In other words, if we pass the mixture through a kiln containing red-hot or white-hot lime, it will then burn and will evolve heat. But, under the laws of "wall action," the burning will be most intense where the gas is in actual contact with the hot solids.

With the above in mind, it is evident that we have it in our power to dilate the flame; to cause it to be as long and as voluminous as we please. All we have to do is to dilute our gas with stack-gases, or to dilute the air for combustion. Stack-gases are better for this purpose than pure nitrogen, even were the latter available, for the reason that the contained carbon dioxide, for chemical reasons, has a combustion-retarding action much greater than that of nitrogen, so that our gas mass

need not be so voluminous. If we dilute our air, we can burn oil or solid fuel, if we like, instead of producer-gas.

Presuming that we are burning producer-gas with air diluted with stack-gases, then, by a simple adjustment of the air-valve and of the stack-gas valve, we can make our flame lengthen and spread out till it fills the kiln. It does not burn at once and give gases rising into the arch; it burns all through the kiln, and it burns most rapidly at the point where it comes into contact with the powdered refractory material of the cement mix. In other words, we are developing a large part of our heat where we want it, in place of developing it all somewhere else and then transferring it to where we want it. And in place of calcining the limestone in an atmosphere of nearly pure carbon dioxide, we are calcining it in an atmosphere of flame-gases, an atmosphere which cannot contain so great a proportion of that gas.

By the separation of the calcining and clinkering steps, we can employ a slow-traveling draft-current through the calcining-kiln, because it is not necessary to expel the products of combustion from a zone of intense heat to make room for fresh fuel. The calcination may be accomplished without regard to the clinkering-zone conditions, and regulation should secure a low temperature of chimney-gases, which, under present practice conditions, carry off approximately six times the heat of the discharged clinker; and practically half of the heat waste accounted for in the chimney-gases is carried by the excess air required to support the combustion of powdered fuel. Prof. Joseph W. Richards[1] reported a test of a rotary kiln showing his calculation of a heat-balance accounting for 72.1 per cent. of the total heat as lost in the chimney-gases.

The heat required for the combination of the clinker is variously estimated, but from 15 to 18 per cent. of the heat developed by the coal seems a fair proportion. Therefore, the clinkering-operation conducted in a separate kiln should require but little fuel and much less attention, as good clinker depends most largely on properly-calcined material.

What I propose for the calcination is the present rotary kiln,

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[1] *Journal of the American Chemical Society,* vol. xxvi., No. 1, pp. 81 to 88 (Jan., 1904).

gas-fired with long-flame combustion. Such a kiln with such a gas-flame has produced more than 6 lb. of lime per pound of coal burned in a producer with low fuel-bed. "Good" gas-fuel reduced the output of the kiln materially. Next, I propose to take fritted calcines from the calcining-kilns to a special clinkering-kiln, which shall embody heat-regenerators and gas-firing. Open-hearth practice gives us an idea of this operation.

This plan has not yet been put into commercial operation. German scientists have thought well of it, and it was expected that a demonstration would take place near Berlin this spring, but from last reports little progress, except on paper, has thus far been made. We can, therefore, only consider theoretically this proposition of cement-burning in a two-kiln operation.

Let us assume the production of a theoretical cement of the following composition, on which heat-determinations will be calculated:

	Per Cent.
Lime, corresponding with 3 (Ca_3SiO_5) and 1 ($Ca_2Al_2O_5$),	68.55
Silica,	20.10
Alumina,	11.35

All of the other constituents found in commercial cement are accidental impurities.

To produce the cement, the ingredients will be used in the following proportions:

	Kg.
Limestone ($CaCO_3$),	122.32
Clay ($Al_2O_3 . 2 SiO_2 . 2 H_2O$),	28.75
Sand,	6 7
Total weight of mixture,	157.77

This amount of raw materials is required to produce 100 kg. of cement.

First, the mix must be heated to 900° C. to dehydrate the clay and decompose the limestone.

The heat-requirements for producing calcines are as follows:

For dehydrating the clay, there will be required 1,218 calories per kilogram of water dissociated from the clay. $2 H_2O =$ 4.01 kilograms.

For dissociating CO_2 from limestone 1,026 calories per kilogram of CO_2 are required. It will be assumed that the specific heat of the mix is about 0.25.

$157.77 - 4.01 = 153.76 \times 0.25 \times 900$, . = 34,596 calories for heating.
$4.01 \times 1,218$, ... = 4,882 calories for dehydrating clay.
122.32 kg. of $CaCO_3$ = 53.76 kg. $CO_2 \times 1,026$ = 55,157 calories for dissociating limestone.

94,635 calories for producing calcines.

For sintering, the heat-requirements are very much less; in fact, the exothermic reactions produce one-third of the heat required for heating the calcines to the sintering-temperature. The heat of combination of lime with silica and alumina in cement does not seem to have been accurately determined, but Le Chatelier found that in the combination of $3CaO, Al_2O_3, 3SiO_2$, 150 calories were developed per unit of $Al_2O_3, 2SiO_2$, and as the silica and alumina exist in about that ratio in the cement, their sum multiplied by 150 calories will give the heat produced.

Clinkering.—For clinkering, estimating that the calcines are discharged directly from the primary kiln into the clinkering-kiln at 900° C., and are therein heated to 1,300° C., a range of 400° rise in temperature, and assuming the specific heat of the calcines to be 0.30, $100 \times 0.3 \times 400 = 12,000$ calories are necessary. Therefore,

$100 \times 0.3 \times 400$ (1,300 — 900) = 12,000 calories absorbed by clinker.
31.45 ($SiO_2 + Al_2O_3$) $\times$ 150, = 4,717 calories produced.

7,283 calories absorbed from fuel.

This represents slightly less than 1 per cent. of fuel for clinkering.

The total heat-units in the clinker as discharged would be:

$$100 \times 0.3 \times 1,300 = 39,000 \text{ calories.}$$

The above calculation represents the heat-requirement, provided that the combustion-gases left the kilns cold, and that the carbon dioxide gas from the limestone left the kiln at the temperature of dissociation.

Calcining.—Practical working-tests with a gas-fired rotary kiln, 100 ft. long by 6 ft. in diameter, burning limestone crushed to 0.5-in. size, such limestone containing 98 per cent. of $CaCO_3$, have shown that it is safe to assume an output of 6 parts of lime to 1 part of good gas-coal; therefore 68.55 kg. of lime

would require 11.43 kg. of fuel, and as there are 68.55 kg. of lime in 100 kg. of cement, 11.43 kg. of coal would calcine all of the lime to produce 100 kg. of cement. There remains only the heat-requirement for bringing 31.45 kg. of $SiO_2 + Al_2O_3$ to the temperature at which the lime is formed. $SiO_2 + Al_2O_3 =$ 31.45 kg. $\times$ 0.25 $\times$ 900 = 7,076 calories, which would be equivalent to not more than 1 kg. of coal per 100 kg. of calcines; therefore, an estimate of work in the calcining-kiln would be:

11.43 kg. of coal for heat to produce 68.55 kg. of lime.
1. kg. of coal for heat to raise temperature of clay.
0.5 kg. of coal allowance for heat-losses in heating up clay.

12.93 kg. of coal per 100 kg. of calcines, or 1 of coal to 7 of calcines.

Assuming 180 tons to be clinkered per day in one clinkering-kiln, 180 tons = 180,000 kg. divided by 100 = 1,800 kg., or 1 per cent. There remain 39,000 calories in the discharged clinker, and assuming that one-third of this heat can be taken up by air-currents and supplied to the primary or calcining-kiln, 13,000 calories would be afforded, equivalent to 1.5 kg. of coal per 100 kg. of calcines, which, subtracted from 12.93 kg. of coal used in calcining, gives 11.43 kg., or 11.43 per cent., of fuel, or 1 of coal to 8.75 of calcines.

Assuming an efficiency of only 25 per cent. in the clinkering-kiln, or 4 kg. of fuel per 100 kg. of calcines, 1,800 kg. $\times$ 4 = 7,200 kg. This would add 4 per cent. to the fuel-consumption, or 4 kg. per 100 to the 11.43 required for calcining = 15.43 kg. per 100, or 15.43 per cent. = 1 of coal to 6.48 of cement.

A regenerative system such as is shown by this plan is capable of utilizing for clinkering 84.2 per cent. of the total heat of the gas produced, and the producer should have an efficiency of 80 per cent., thus giving 67.36 per cent. efficiency for the combination of the producer and kiln, not allowing for loss by radiation. I believe it would be safe to assume 50 per cent. efficiency for the clinkering-kiln outfit, allowing 17.36 per cent. for loss by radiation. If this result should be realized, the process would yield 7.44 kg. of cement for each kilogram of coal consumed, based on practical working-tests with the gas-fired rotary kiln burning limestone crushed to pass a 0.5-in. ring.

Field-Investigations of Structural Materials by the U. S. Geological Survey.

BY ERNEST F. BURCHARD, WASHINGTON, D. C.

(Pittsburg Meeting, March. 1910.)

IN connection with the work of testing structural materials for the use of the U. S. government at the laboratories of the technologic branch of the U. S. Geological Survey at St. Louis, Mo., from September, 1904, to April, 1909, and since then at Pittsburg, Pa., large quantities of sand, gravel, and crushed stone have been needed from time to time, besides samples of building-stone, clays, and other quarry-products. In many instances it has been necessary to obtain single samples of sand, gravel, or crushed stone in lots of a car-load or more. For obvious reasons all these collections have had to be made by a representative of the Survey. As the work progressed it was found essential that numerous geologic data should be gathered regarding the materials collected, in order that they might be properly classified as to kind of material, source, geologic age, etc., and that they might be rated according to quantity available, accessibility to transportation facilities, approximate costs, suitability for special purposes, etc., and that notes might be taken regarding undeveloped deposits of possible value. Furthermore, it was desired that the selection of materials for tests should be made broadly, so as to cover, first, the representative types of material, and later, if possible, the typical materials of the more important building-centers in the United States.

In order to insure a certain degree of uniformity in this work and economy of effort and expense, it was decided that as much of the sampling as was practicable should be done by a member of the geologic branch, whose regular field-trips could be combined conveniently with those necessary for procuring materials for the laboratory. During parts of the last three years I have been engaged in field-work of this sort.

The results of the field- and laboratory-investigations of structural materials are intended primarily for the information of those departments of the government which are engaged in construction-work—namely, the office of the Supervising Architect of the Treasury Department; the War Department; the Isthmian Canal Commission; the Reclamation Service of the Interior Department; and the Navy Department. The field- and test-data have been made available to these departments, although comparatively little of the information has yet been published. However, when fairly complete information has been obtained that is thought to be of general interest, it has been compiled and published in bulletins by the Geological Survey.

Prior to the fiscal year 1909–1910 the making of general typical collections and of special areal studies was the principal work undertaken. In the spring of 1909 a request for information was received by the Director of the Survey, through the Secretary of the Interior, from the Secretary of the Treasury, accompanied by a list of about 380 cities in which the construction of new Federal buildings and extensions had been authorized by Congress and would be in progress in various stages within the next three or four years. This work will involve an expenditure of not less than $50,000,000. The following quotations from this request are therefore of interest here:

"It will be in the interest of the Government to have the Geological Survey furnish this Department with information as to the character and extent of the sands, gravels, stone, and other materials suitable for work in concrete construction; sands for mortars; stone for masonry work; brick, terra cotta, hollow tile, and other structural materials that may be suitable and available for use in the construction of these buildings within easy reach of each of the localities named in this list. Exact and thorough data of this kind will not only be of great service in connection with the construction of the buildings in question, but also in the preliminary planning of them."

"Reports should be submitted to the Supervising Architect's office from time to time, as rapidly as any part of this work can be completed, or definite results obtained, without awaiting publication, in order that this information may be available for use in the planning and construction of public buildings already authorized."

In view of the urgency of this request, it became necessary to make this work a special order of business, rather than an incident to other field-trips. Therefore, during the second half

of 1909 part of the time of six geologists, J. A. Holmes, T. Nelson Dale, N. H. Darton, J. E. Todd, James A. Gardner, and J. A. Udden, and all of my time was spent on this work, with the result that about 85 per cent. of the 380 specified localities, including all where the need was urgent, were visited and a field-report on each one was prepared and sent to the Supervising Architect. At the outset it was planned that there should be uniformity in the reports, and therefore they were prepared mostly according to the following outline, care being taken that only data of a practical character should be given:

Structural Materials Investigated for Use in Federal Buildings.

I. *Stone:*

- A. Dimension stone for exterior.
  - *a.* Foundations.
  - *b.* Walls.
  - *c.* Sills and trim.
- B. Ornamental stone for interior (marble, serpentine, onyx, etc.).
- C. Slate for roofing, sanitary fixtures, etc.

II. *Material for concrete:*

- A. Sand.
- B. Gravel.
- C. Crushed stone, slag, cinder, shell, etc.
- D. Cement—Portland, natural, hydraulic, etc.

III. *Clay products:*

- A. Brick.
  - *a.* Common.
  - *b.* Front (pressed, rough, fire-faced, etc.).
- B. Tile.
  - *a.* Roofing.
  - *b.* Hollow building-tile or block.

IV. *Materials for mortars and plasters:*

- A. Lime.
  - *a.* Quick.
  - *b.* Hydrated.
- B. Gypsum wall-plasters.
- C. Sand.

The work was necessarily done very rapidly, and the reports were not written for geologists, but rather for the use of persons who may not have had training in geology. The view-points of the geologist, however, respecting extent, quality, structure, and general geologic relations that affected the utilization of material were borne in mind throughout the work. The reports as type-written range in length from 1 to 12 pages but average less than 5 pages.

In making these field-studies it was the endeavor to relieve

the laboratory of all work possible and to give to the Supervising Architect a definite opinion as to the value of the material, backed up by a detailed description of it and the results of field-tests. In addition, any authentic test-data in possession of the producer or contained in State geological survey reports were utilized. The common points considered with regard to stone, gravel, clay, gypsum, etc., were noted on forms in loose-leaf books. Many special details with regard to the various materials had to be investigated, and for brick and other clay-products notes were kept regarding the processes of manufacture. Sands were subjected to qualitative tests for the presence of lime, alkali, clay, magnetite, quicksand, and silt. Granular-metric analyses were made and the material was critically examined under the field-lens. A knowledge of the architect's general specifications was requisite, and as a rule the geologist was able to tell, after a careful investigation, whether a sand, gravel, stone, or brick would fulfill these specifications or not.

To the question as to what direct advantage these reports would be to the Supervising Architect, or to federal construction-work in general, the following answer may be given:

(*a*) Attention is called to materials of merit which owing to their proximity to the building site should be obtainable at lower prices than similar materials occurring long distances away.

(*b*) Attention is called to little-developed and hitherto comparatively unknown materials that may possess special merit for certain kinds of work.

(*c*) Warning is issued against the use of materials that are not suitable yet are commonly used in certain localities.

(*d*) Warning is issued against the acceptance of materials from deposits which may be of good quality, but of insufficient quantity.

(*e*) Warning is issued against the acceptance of materials from deposits which may afford small samples of material of excellent grade but whose quality as a whole is inferior.

(*f*) Data regarding local costs and freight-rates are obtained on small and large lots of all materials shipped into the locality, such as cement, stone, sand, wall-plasters, etc., thus affording aid towards the preparation of specifications for new buildings.

In addition to the results of this work as related to the gov-

ernment, its relation to the country at large may be mentioned. Where little-known but meritorious materials are thus brought to the attention of the Supervising Architect, and incidentally to that of the public, as they doubtless will be later through their use in federal buildings and their description in published reports, the utilization of important natural resources is encouraged. During the past field-season many materials that would probably otherwise have been passed unnoticed have been brought to the attention of the Supervising Architect.

Besides the work outlined above, there have been at times during the past year at the laboratories of the Survey, both in Pittsburg and in Washington, special investigations of such subjects as the manufacture of hydrated lime and studies of Keene's cement and wall-plasters, tending towards the formulation of standard specifications for these materials in government construction-work. Here again the services of field-geologists have been required to procure the samples and the geologic data of the Survey thereby enriched.

Heats of Formation of Some Ferro-Calcic Silicates.

BY H. O. HOFMAN AND C. Y. WEN, MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON, MASS.

(Pittsburg Meeting, March, 1910.)

I. Introduction.

In casting a thermal balance of the heat generated and absorbed in a blast-furnace treating lead-, copper- and similar non-ferrous ores, assumptions have always to be made for the values of the heat of formation of slag. Considering that the larger part of the charge introduced into the furnace comes out in the form of slag, the significance becomes apparent of a lack of heat-values for the singulo-silicate, which is the type of slag commonly made. The aim of the present investigation is to supply these heat-values as far as possible with the apparatus and the time available.

The heats of formation of the ferrous and manganous bisilicates have been determined by Le Chatelier;[1] those of some silicates other than iron and manganese by Tschernobaeff;[2] finally Richards[3] gives a figure for a copper blast-furnace slag of known composition. The existing data and those resulting from the present investigation are assembled in Table VI.

II. Materials Used.

The materials used in the experiments were silica, ferrous oxide, and calcium carbonate.

Silica.—The silica sold in the market as chemically pure usually contains small amounts of magnesia, lime, and alumina, and is therefore not suited for the work. The pure material was obtained from clear quartz crystals from Arkansas, broken in a diamond mortar and ground in an agate mortar to pass

[1] *Comptes rendus*, vol. cxx., p. 623 (1895); vol. cxxii., p. 80 (1896).

[2] *Revue de Métallurgie*, vol. ii., p. 729 (1905); *Electrochemical and Metallurgical Industry*, vol. iv., No. 2, p. 72 (Feb., 1906).

[3] *Electrochemical and Metallurgical Industry*, vol. v., No. 7, p. 266 (July, 1907); *Metallurgical Calculations*, vol. iii., pp. 474, 475 (1908).

through bolting-cloth. The fine powder was treated several times with concentrated hydrochloric acid, washed and dried. Two determinations, using the method of volatilization as silicon fluoride, gave SiO_2, 99.98 and 99.94, average 99.96 per cent.

Ferrous Oxide.—Passing by the older unsatisfactory methods of a partial reduction of ferric oxide by means of hydrogen, carbon monoxide, and mixtures of carbon monoxide and dioxide, as carried out by Wackerwoder-Stromyer,[4] Debray,[5] and Moissan;[6] as well as the attempts of Vogel,[7] Siewert,[8] Moissan,[9] and Birnie[10] to obtain ferrous oxide by the decomposition of ferrous oxalate, which does not furnish pure material; there remain the methods of Tissandier[11] of oxidizing metallic iron at elevated temperature in a current of carbon dioxide, and of Berthier,[12] and Plattner,[13] of heating together metallic iron and hammer-scale.

Considerable time was given to the preparation of ferrous oxide by Tissandier's method. Iron powder was spread in a thin layer in a porcelain boat, and this heated in an electric-resistance furnace, first in a current of hydrogen for 45 min., and then cooled for 10 min.; subsequently carbon dioxide was turned on, the temperature raised to the desired degree, held for a stated time, and the sample finally allowed to cool in the gas-current. It was found that, in order to convert most of the Fe into FeO with a sample weighing 0.5 g. spread thinly in a boat 2.5 by $\frac{7}{16}$ in., heating for 3 hr. at a temperature of 680° C. was necessary, the current of carbon dioxide passing through a wash-bottle at the rate of from 100 to 120 bubbles per minute. At 700° C. some of the Fe was converted into F_3O_4. In spite of numerous trials no manner of operation was obtained which insured the absence either of Fe or Fe_3O_4. It

[4] *Archiv für Pharmacie*, vol. lxxxvi., p. 27 (1843).
[5] *Comptes rendus*, vol. xlv., p. 1018 (1857).
[6] *Annales de Chimie et de Physique*, Fifth Series, vol. xxi., p. 199 (1880).
[7] *Liebig's Annalen der Chemie und Pharmacie*, vol. xcv., p. 116 (1855).
[8] *Jahresberichte über die Fortschritte der Chemie*, vol. xvii., p. 265 (1864).
[9] *Op. cit.*
[10] *Trans.*, xxix., 684, foot-note (1899).
[11] *Comptes rendus*, vol. lxxiv., p. 531 (1872).
[12] *Traité des Essais*, vol. i., p. 445; vol. ii., pp. 186, 188 (1834).
[13] Merbach, F. T., *Die Anwendung der Erwärmten Gebläseluft in der Metallurgie*, p. 309 (1840).

was found later that Moissan [14] and H. Steffe [15] had been equally unsuccessful. Berthier [16] prepared ferrous silicates by heating in a clay crucible a mixture of iron filings, hammer-scale, and silica. He says [17] that iron filings and ferric oxide heated together form "oxide de battiture," a compound of the composition of hammer-scale; also that this "oxide de battiture" is not reduced to FeO by heating with metallic iron. However, Mills [18] patented a process for preparing ferrous oxide from ferric oxide and metallic iron by heating a mixture of the proportions required by the formula $Fe_2O_3 + Fe = 3\ FeO$, and H. Steffe [19] found the method satisfactory in his investigations. It was therefore adopted for the preparation of ferrous oxide for the experiments, and met all the requirements.

The metallic iron used was the purest iron powder of Kahlbaum. Two analyses gave, Fe, 99.10 and 99.08 per cent. The ferric oxide was the purest brand made by the same firm; an average of three analyses gave, Fe, 69.70 per cent., corresponding to Fe_2O_3, 99.57 per cent. The nitrogen required to furnish the neutral atmosphere in which the mixture of metallic iron and ferric oxide was to be heated was prepared by conducting a mixture of air and ammonia gas over cupric oxide heated in a combustion-tube. The apparatus used for the preparation of ferrous oxide is shown in Fig. 1, in which *A* is a 500-cc. flask containing ammonia, sp. gr. 0.90; *K*, combustion-tube, 3 ft. long by 0.5 in. in diameter, charged with cupric oxide; *B*, gas-heated combustion-furnace; *C*, 250-cc. flasks charged with potassium pyrogallate to absorb oxygen; *D*, 250-cc. flasks containing potassium hydroxide to absorb carbon dioxide; *E*, a 250-cc. flask containing an ammoniacal solution of cuprous chloride to absorb any carbon monoxide present; *F*, 250-cc. flasks containing concentrated sulphuric acid to remove moisture and thus dry the purified gas before it enters *H*, the electric-resistance furnace. The nitrogen thus prepared still retained a small quantity of oxygen, which became noticeable

[14] *Annales de Chimie et de Physique*, Fifth Series, vol. xxi., p. 199 (1880).
[15] Doctorate thesis, Berlin, p. 7 (1908).
[16] *Traité des Essais*, vol. i., p. 445.
[17] *Idem*, vol. ii., pp. 186, 188.
[18] German patent, Klasse xii., No. 86,589 (Mar. 31, 1895).
[19] *Op. cit.*, p. 8.

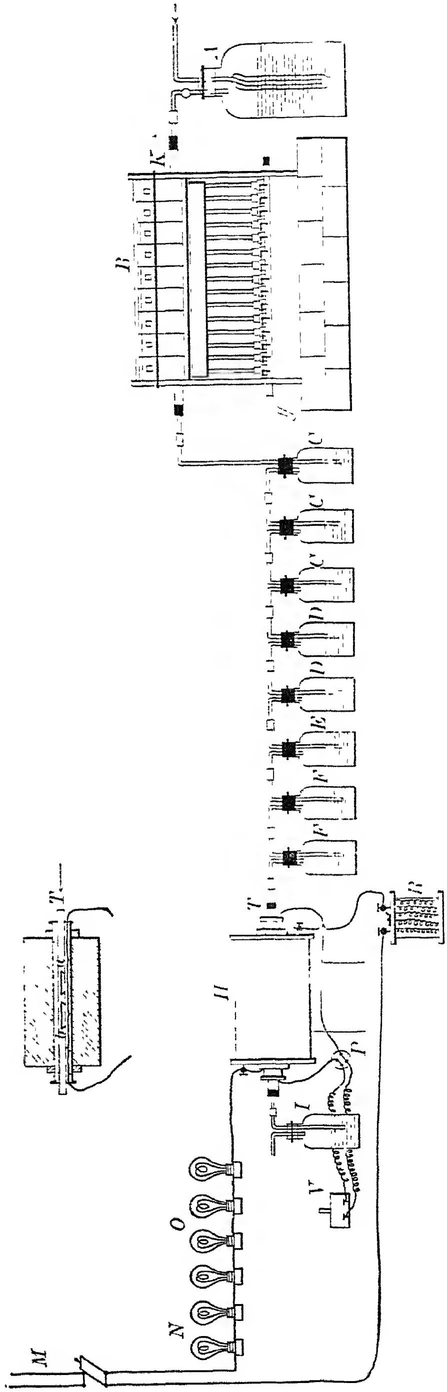

Fig. 1.—Arrangement of Apparatus for Preparation of Ferrous Oxide.

by its effect upon the charge, but could not be detected by the usual method of analysis with the Orsat apparatus. In order to prevent all oxidation of the charge, a boat, *a*, filled with iron powder was placed in the heating-tube, *T*, near the end through which the nitrogen was to enter, in order that the iron might combine with the trace of oxygen contained in the nitrogen, and thus protect the tube, *b*, filled with the charge. *H* is an electric-resistance furnace, 12 in. long by 8 in. in diameter; *T*, a porcelain combustion-tube, 22 in. long by $\frac{5}{8}$ in. in diameter; *I*, a sealing-bottle charged with sulphuric acid; *M*, a 110-volt circuit with switch; *N*, two 100-c-p. and *O*, four 32-c-p. incandescent lamps connected in parallel; *R*, a rheostat; *P*, the cold-junction of the thermo-electric pyrometer; and *V*, a millivoltmeter.

In making up the mixture of iron powder and ferric oxide, due consideration was given to the percentage of iron in each substance. The iron powder was ground in an agate mortar to pass a bolting-cloth. The components of a charge were rubbed together in the mortar for 15 min. so as to obtain an intimate contact of particles, and then transferred into a boat, 3 in. long by $\frac{3}{8}$ in. wide by 0.25 in. deep, made of thin Russia iron. The boat was then introduced into the furnace and followed by another filled with metallic iron. After closing the end of the tube, the gas in the combustion-furnace was turned on, and the current of nitrogen passed through the resistor furnace at a rate indicated by not more than 6 nor less then 3 bubbles per minute in the wash-bottle. When the air in the electric furnace had been replaced by nitrogen, the current of 10 amperes at 110 volts was turned on. It raised the temperature of the tube to about 800° C. in from 2.5 to 3 hr. The charge was held for 2 hr. at a temperature ranging from 820° to 840° C., the current shut off, the charge allowed to cool in the current of nitrogen, and the gas and the air-current in the combustion-furnace stopped. Pure FeO contains, Fe, 77.70 per cent. In testing a treated charge, the total iron was determined. The presence of any metallic iron was ascertained by boiling a small sample in a test-tube with distilled water, to drive off the air, and then adding a few drops of dilute hydrochloric acid, when small bubbles of hydrogen would indicate its presence. The test for Fe_2O_3 consisted in heating the sub-

stance in dilute hydrochloric acid with the exclusion of air. The latter was expelled by passing a current of carbon dioxide through the tube while the sample was being dissolved. The presence of even a very small amount of Fe_2O_3 is easily recognized by a yellowish color. The results obtained are recorded in Table I.

TABLE I.—*Preparation of Ferrous Oxide.*

No of Test.	Charge.	Metallic Iron.	Ferric Oxide	Temperature.	Time.	Product, Fe.	Remarks
	Grams.	Grams.	Grams.	Degrees C.	Hrs.	Per Cent.	
1	3.8436	1.0000	2.8436	600	2	74.40	Much Fe and Fe_2O_3.
2	1.9218	0.5000	1.4218	600	2	n. d.	Much Fe and Fe_2O_3.
3	1.9218	0.5000	1.4218	750	2	n. d.	Much Fe and Fe_2O_3.
4	1.9218	0.5000	1.4218	820–840	2	76.70	Little Fe, much Fe_2O_3.
[a]5	3.8507	1.0000	2.8507	820–840	2	77.05	Little Fe, some Fe_2O_3.
[b]6	3.8436	1.0000	2.8436	820–840	2	77.30	Trace Fe, some Fe_2O_3.
7	3.8250	1.0000	2.8250	820–840	2	77.67	Trace Fe, no Fe_2O_3.
8	3.8250	1.0000	2.8250	820–840	2	77.68	Trace Fe, no Fe_2O_3.
9	3.8250	1.0000	2.8250	820–840	2	77.62	Trace Fe, no Fe_2O_3.
10	3.8250	1.0000	2.8250	820–840	2	77.65	Trace Fe, no Fe_2O_3.
11	5.7375	1.5000	4.2375	820–840	2	77.83	Trace Fe, no Fe_2O_3.

[a] The charge contained an amount of Fe_2O_3 sufficient to allow for the small percentage of H in the iron powder.

[b] The protecting boat charged with iron powder was used here for the first time.

Tests Nos. 1, 2, and 3 show that the temperature was too low; an improvement is seen in test No. 4, in which the temperature has been raised; No. 5 discloses a higher percentage of iron with increase in the weight of the charge; the introduction, with test No. 6, of the boat filled with iron powder in front of the boat containing the charge, gives a decidedly better result; reduction of the amount of Fe_2O_3 in test No. 7 raises the iron-content close to the theoretical figure of 77.7 per cent., while Fe_2O_3 is absent, and only a trace of Fe can be detected. The ferrous oxide for the experiments was prepared as shown by tests Nos. 7 to 11.

Calcium Carbonate.—The precipitated salt, sold as chemically pure, was used. It ought to contain 56.06 per cent. of CaO; chemical analysis gave 55.85 per cent.

III. METHOD OF DETERMINING THE HEAT OF FORMATION OF SLAG.

The method of determining the heat of formation followed by Le Chatelier, Tschernobaeff, and Richards is the one devised

by Le Chatelier.[20] It consists in mixing the slag-forming constituents with sufficient finely-divided charcoal to raise the temperature of the charge well above the formation-temperature of the slag when the charcoal is burned in oxygen in a calorimetric bomb. The difference between the pre-determined calorific power of the charcoal and that of the charge gives the

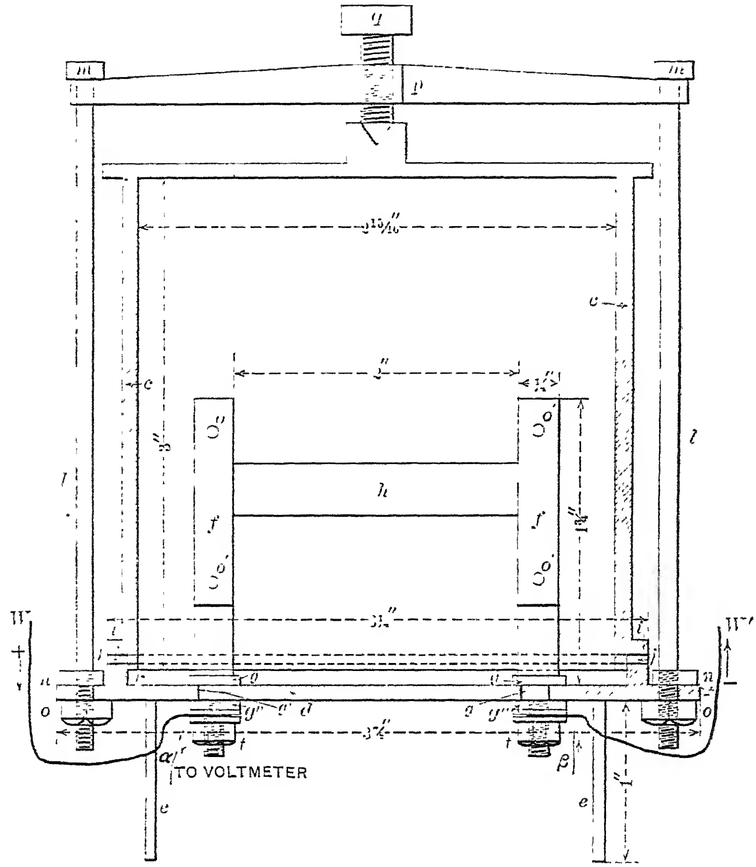

Fig. 2.—Vertical Section of Bomb.

quantity of heat generated or absorbed in the formation of the slag. In view of the danger of ferrous oxide being reduced by carbon when the charge contains considerable amounts of the former, especially in the presence of lime, it seemed advisable to try another means of heating in which carbon was ex-

[20] *Comptes rendus*, vol. cxx., p. 623 (1895).

cluded. This was found in fusing the charge in a platinum boat by means of electricity, the boat to be contained in a bomb, and the latter to be submerged in a calorimeter. The difference between the quantity of heat developed by the electric energy alone and the quantity developed when fusing the charge gives the heat of formation of the silicate or the silicate mixture.

The Bomb.—The bomb, constructed for experiments on heats of combustion for fuel by Prof. C. L. Norton, is shown in vertical section in Fig. 2. It consists of an inverted cylindrical vessel, *c*, and a bottom-plate, *d*, with tripod, *e*; both are of rolled brass treated with a solution of cupric carbonate in ammonia, which gives it a dark surface not attacked by any corroding influences of the calorimeter-water. The cylinder has on the bottom a seat for the set-screw, *q*, and on the top the collar, *i*; the bottom has near the circumference a rib, *k*; a rubber gasket, *j*, makes a gas-tight joint when cylinder and plate are pressed against one another by means of rods, *l*, the cross-piece, *p*, and the set-screw, *q*. A rod, held to the bottom-plate by the tightening-nut, *o*, and the lock-nut, *n*, is loosely connected with the cross-piece by the head, *m*. The platinum boat, *h*, 1.75 in. long by $\frac{5}{16}$ in. high and $\frac{1}{8}$ in. wide, made of foil 0.02 mm. thick, is held by two slotted brass poles, *f*, each provided with tightening-screws, o'; a pole, $2\frac{3}{16}$ in. long, is 0.25 in. in diameter for $1\frac{11}{16}$ in. and $\frac{1}{8}$ in. in diameter for the remaining 0.5 in. It passes through the bottom-plate, resting with its shoulder upon an insulating rubber washer, *g*, and is firmly connected with it by means of the tightening-nut, *t*; rubber washers, g' and g'', prevent metallic contact. Between the washer, g'', and the nut, *t*, electric connection is made through the wires, *W* and W', with the storage-battery which furnishes the heating-current, and through wires, α and β, with the voltmeter.

Calorimetric and Electric Apparatus.—Their arrangement is shown in Fig. 3. The calorimeter, *D*, is a cylindrical nickel-plated brass vessel, 6 in. high and 5 in. in diameter. It holds the bomb, *C*, thermometer, *T*, permitting readings to 0.01° C., and the electrically-driven propeller-stirrer, *S*. The bottom is provided with cork legs on which it stands in a varnished paper-pulp bucket, *J*, 8.5 in. high and 7.75 in. in diameter, inclosed

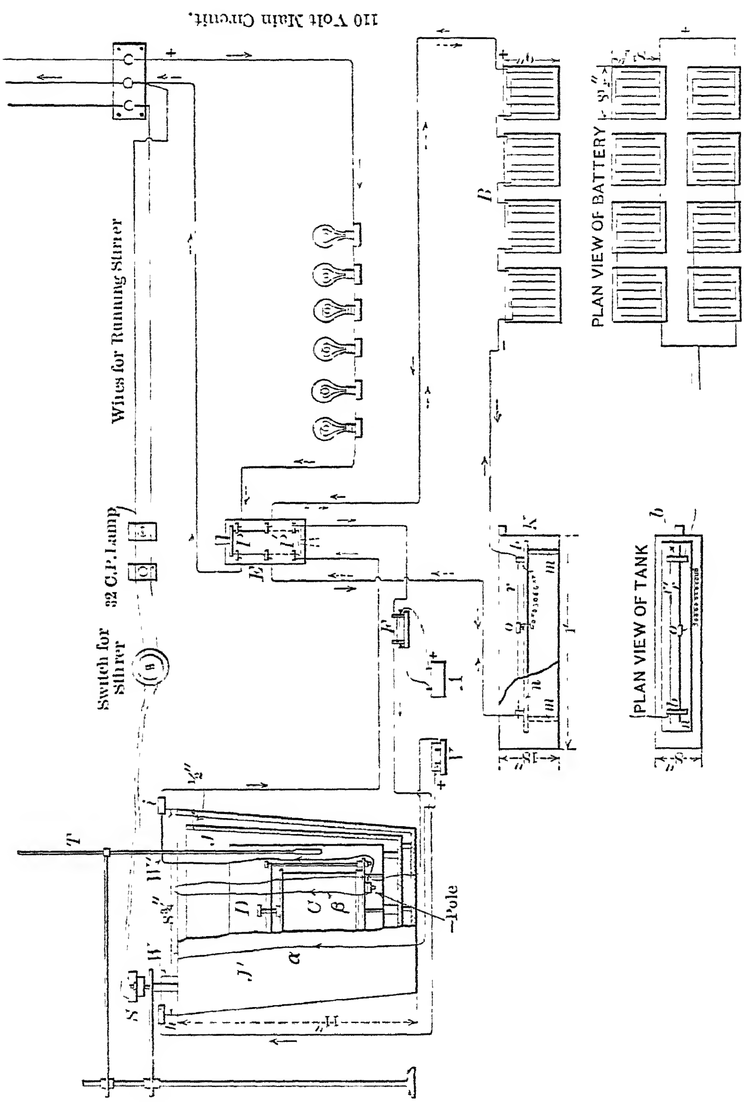

FIG. 3.—ARRANGEMENT OF CALORIMETRIC AND ELECTRIC APPARATUS.

by a second bucket, J', 11 in. high and 8.75 in. in diameter, of the same material.

The current for fusing a charge was supplied from a storage-battery, B, of eight cells, four connected in series, and the two series in parallel. Each cell has 11 electrodes, 8 by 8 in., and furnished a current of 25 amperes at 2 volts. The strength of the current was measured by the ammeter, A, connected in parallel with the shunt, F, which is in series with the circuit; the instrument permits readings to 0.1 ampere. The difference in potential was measured by the voltmeter, V, permitting readings to 0.001 volt. The storage-battery was kept charged from the 110-volt main circuit: not more than 27 amperes was drawn from each cell. In charging the battery, the current from the supply-main traveled, in the direction indicated by the dotted arrows, through six 32-c-p. incandescent lamps, the switch in the position P, to the battery, B, returned through the resistance-wires, r, r', back to P, and the return-main. In discharging the battery the switch was thrown into position P'. The current, leaving the battery, follows the full-drawn arrows, passes through the right side of P', the shunt, F, the connector, h, and enters the bomb at the bottom through the positive pole, W, Fig. 2, passes through the platinum boat, leaves the bomb at the bottom through the negative pole, W', passes through the connector, i, the left side of the switch, P', the two resistance-wires, r, r', and returns to the battery.

The copper conducting-wires, of No. 8 B. & S. gauge, are thick enough to conduct the current without becoming heated. For the external resistance, iron wire was at first tried; but proving unsatisfactory it was replaced by No. 14 I_a I_a wire of the Herman Boker Co., Brooklyn, N. Y., which offered a resistance of only 73 ohms per 1,000 ft. Two wires, r, r', 3.5 ft. long, were connected in parallel by screwing into the brass blocks, b, 0.5 in. sq. and 4 in. long, which were fastened to the board, n, resting on the bricks, m; the whole was immersed in the water-tank, K. As the platinum becomes heated by the passage of the current, its resistance will increase and the current correspondingly decrease. In order to compensate for this, the slide, o, was used with the wire, r, and the current kept constant by increasing or decreasing the external resistance.

Method of Operation.—In the experiments, every determination with a charge was preceded by duplicate blank-tests: and in every test the current, the quantity of water (1,200 g.), and the time (1 min.) were kept constant.

The temperature of the calorimeter-water was so adjusted that at the end of a test it should be about the same as the room-temperature, because in this way the cooling-correction was reduced to a minimum. Usually, the calorimeter-water at the start was 1° C. lower than the room-temperature. Care was had in introducing the charge, after the blank-tests had been made, to leave the position of the boat absolutely unchanged. When the bomb had been closed, it was placed in the calorimeter charged with the standard quantity, 1,200 g. of water, the stirrer set in motion, the thermometer lowered into the water, and connection made with the voltmeter. The stirrer was allowed to run for not less than 5 min., usually 10 min., before the preliminary readings of the temperature of the water were taken; observations were then made every minute for 5 min., when the current was turned on and allowed to pass accurately 1 min., which was timed by a stop-watch. The temperature of the water would rise and continue to do so for 3 or 4 min. more before it reached its maximum. After the current was turned off, readings were again begun and taken every 30 sec., for 5 min. after the temperature of the water had reached its highest point.

During the passage of the current, the ammeter and voltmeter were closely watched, and the external resistance was varied by means of the slide, *o*, Fig. 3, to keep the total current constant. It may be observed that the whole current did not pass through the platinum boat, but that a small part went through the voltmeter, *V*. Owing, however, to the high resistance of the voltmeter and the low voltage of the storage-battery, only an inappreciable part of the total current passed through the voltmeter, and could be, therefore, neglected.

The record of one of the two blank-tests made for determining the heat of formation of 1 g. of ferrous singulo-silicate (Table III., Test 1) may serve as an example of the method followed in making observations and in computing the cooling-correction.

TABLE II.—*Record of Blank-Test 1a.*

Duration of Time.		Temperature Calorimeter-water.	Temperature of Room, Average	Electric Energy.	
Min	Sec.	Deg. C.	Deg C.	Amp	Volts
0	0				
	30	22.05	23.5	50	4.100
1	0				
	30	22.05	....	......	4.125
2	0			......	4.125
	30	22.06	....	......	4.115
3	0				
	30	22.06	....	......	4.150
4	0			......	4.100
	30	22.07	....	......	4.125
5 [a]	0				
	30	22.07	....	......	4.125
6 [b]	0			......	4.125
	30	22.14	....	......	4.125
7	0	24.25	....		
	30	24.30	....	......	4.125
8	0	24.32	....	......	4.115
	30	24.35	....	......	4.110
9	0	24.37 max.	....		
	30	24.37	....	......	4.110
10	0	24.37	....	......	4.110
	30	24.37	....	......	4.100
11	0	24.37	....		
	30	24.37	....	......	4.115
12	0	24.36	....	......	4.115
	30	24.36	....	......	4.110
13	0	24.36	....		
	30	24.35	....	......	4.100
14	0	24.34	....		
	30	24.34	....	Average,	4.116

a Current turned on.
b Current turned off.

While the experiment is being carried on, the calorimeter may absorb heat from the room or radiate heat into it. In order to remedy this, a correction has to be applied, and this is calculated from the formula

$$T = (t_2 - t_1) + \frac{(r_1 + r_2)}{2} \theta_2 - \theta_1).$$

Here t_1 denotes the temperature in C.° at the moment when the current is turned on; t_2, the maximum temperature after the current has been turned off; r_1, the rate of radiation per minute in C.° before the current is turned on; r_2, the rate after the current has been turned off; θ_1, the time the current was turned on: θ_2, the time at which the temperature reached its maximum. If r_1 is an increasing rate, *i.e.*, if the temperature

changes, as in the above experiment, from 22.05° to 22.07°, heat is given to the calorimeter and its contents from the surroundings, and r_1 must have a negative sign. Similarly, r_2, which is found in the example to show a decrease from 24.37° to 24.34°, must have a positive sign. The expression $\frac{(r_1 + r_2)}{2}(\theta_2 - \theta_1)$ may be additive or subtractive according to the signs of r_1 and r_2. In the experiment under consideration, $t_1 = 22.07$: $t_2 = 24.37$: $\theta_1 = 5$ min.: $\theta_2 = 9$ min.: $r_1 = \frac{22.07 - 22.05}{5} = -0.004$: $r_2 = \frac{24.37 - 24.34}{5} = +0.006$: the cooling correction $= \frac{(-0.004 + 0.006)}{2} 4 = +0.004$: $T = 24.37 - 22.07 + 0.004 = 2.304°$ C.

In making the fusion-tests, as already stated, the theoretical quantity of heat developed by electric energy is compared with the actual quantity obtained in the calorimeter. In the blank-tests the theoretical quantity should be the same as the actual: however, the actual was found to be greater in almost every case, owing to the constant error of the instruments, to the cooling-correction, and to the personal factor in making readings. The theoretical quantity of heat developed by electric energy is $H = 0.2387\ I\ E\ t$ g-cal., in which $I =$ electric current in amperes, $E =$ difference in potential at the poles of the bomb in volts, $t =$ time in seconds. The actual quantity of heat, $H_1 = Q\ T$, in which Q denotes the quantity of water in grams, including the water-equivalent of the calorimeter, the stirrer, and the bomb, and T the corrected rise in temperature. The water-equivalent, obtained by multiplying the sum of the weights of the calorimeter, 211 g., stirrer, 100 g., and bomb, 836 g., by the specific heat of the metal parts, is $(211 + 100 + 836) \times 0.09 = 103$ g. of water. With 1,200 g. of water, $Q = 1{,}200 + 103 = 1{,}303$ g., and $H_1 = Q \times T = 1{,}303 \times T$.

IV. Results.

Heat of Formation of Ferrous Singulo-Silicate (FeO, 70.80; SiO_2, 29.20 *Per Cent.*).—Table III. gives the results of five charge-tests, each preceded by two blank-tests. Eliminating test No. 1, on account of the charge not having been completely fused, and No. 2 on account of some unknown irregularity, there remain

tests Nos. 3, 4, and 5, which give the values 111, 103, and 115 g-cal. evolved in the formation of 1 g. of ferrous singulo-silicate, with an average of 109 g-cal.

TABLE III.—*Heat of Formation of Ferrous Singulo-Silicate* (*FeO*, 70.80; *SiO_2*, 29.20 *Per Cent.*).

(Total water-equivalent = 1,303 g.; time = 60 sec.; weight of charge = 1 g.)

Blank-Tests.

Test No.	Current.	Difference in Potential.	Room-Temperature.	Corrected Rise in Temperature.	Actual Amount of Heat.	Theoretical Amount of Heat.	Difference, Actual and Theoretical Amounts of Heat.	
							Real.	Average.
	Amperes	Volts.	Deg. C.	Deg C.	G-Cal.	G-Cal	G-Cal.	G-Cal.
1 a.	50	4.116	23.5	2.304	3,002	2,948	54	60
1 b.	50	4.125	24	2.318	3,020	2,954	66	
2 a.	50	3.870	26.5	2.188	2,851	2,771	80	83
2 b.	50	3.862	25.5	2.189	2,852	2,765	87	
3 a.	52	4.042	25	2.381	3,103	3,010	93	90
3 b.	52	4.007	25	2.356	3,070	2,984	86	
4 a.	52	4.082	23	2.387	3,110	3,040	70	72
4 b.	52	4.040	25	2.365	3,082	3,009	73	
5 a.	53	3.851	24	2.310	3,010	2,923	77	80
5 b.	53	3.822	25.5	2.290	2,984	2,901	83	

Charge-Tests.

Test No.	Current.	Difference in Potential.	Room-Temperature.	Corrected Rise in Temperature.	Actual Amount of Heat.	Theoretical Amount of Heat.	Real.	Average.
1	50	4.014	24	2.330	3,036	2,874	162	...
2	50	3.702	24	2.200	2,867	2,651	216	...
3	52	3.911	25	2.390	3,114	2,913	201	...
4	52	4.038	25	2.442	3,182	3,007	175	...
5	53	3.641	23.5	2.271	2,959	2,764	195	...

End-Result.

	Charge-Tests.	Blank-Tests.	Difference, Heat Evolved in Slag-Formation.
	G-Cal.	G-Cal.	G-Cal.
1	162	60	+ 102
2	216	83	+ 133[a]
3	201	90	+ 111
4	175	72	+ 103
5	195	80	+ 115
Average,			+ 109

[a] Omitted.

Heat of Formation of Ferro-Calcic Singulo-Silicate (*FeO*, 57.58; *CaO*, 12.00; *SiO_2*, 30.42 *Per Cent.*).—Table IV. gives the experimental data. Calcium carbonate was used to furnish the lime instead of calcium oxide, as the latter readily absorbs

moisture and carbon dioxide. To furnish 0.12 g. of CaO, there was required 0.2141 g. of $CaCO_3$: the quantity of heat necessary to split this into CaO and CO_2 is 96 g-cal. This amount has been added to the sum of differences between the actual and theoretical amounts of heat obtained in the charge-tests and the average of those of the blank-tests. The tests are sufficiently close to allow averaging them. The heat of formation of 1 g. of this mixture is therefore 140 g-cal.

TABLE IV.—*Heat of Formation of Ferro-Calcic Singulo-Silicate (FeO, 57.58; CaO, 12.00; SiO_2, 30.42 Per Cent.).*

(Total water-equivalent = 1,303 g.; time = 60 sec.; weight of charge = 1 g.)

Blank-Tests.

Test No.	Current	Difference in Potential.	Room-Temperature.	Corrected Rise in Temperature.	Actual Amount of Heat.	Theoretical Amount of Heat.	Difference, Actual and Theoretical Amounts of Heat: Real.	Difference, Actual and Theoretical Amounts of Heat: Average.
	Amperes.	Volts	Deg. C.	Deg. C.	G-Cal.	G-Cal.	G-Cal.	G-Cal.
6 *a.*	53	3.975	22	2.38	3,101	3,017	84	65
6 *b.*	53	4.060	23	2.40	3,127	3,082	45	
7 *a.*	58	3.681	24	2.41	3,140	3,057	83	87
7 *b.*	58	3.702	25	2.43	3,167	3,075	92	
8 *a.*	57	3.681	23	2.37	3,088	3,005	83	82
8 *b.*	57	3.700	23	2.38	3,101	3,020	81	
9 *a.*	59	3.607	24	2.39	3,114	3,047	67	66
9 *b.*	59	3.594	24	2.38	3,101	3,036	65	
10 *a.*	58	3.611	23	2.375	3,095	3,000	95	101
10 *b.*	58	3.613	23	2.385	3,108	3,001	107	
11 *a.*	57	3.694	23	2.37	3,088	3,015	73	79
11 *b.*	57	3.711	23	2.39	3,114	3,029	85	

Charge-Tests.

Test No.	Current	Difference in Potential.	Room-Temperature.	Corrected Rise in Temperature.	Actual Amount of Heat.	Theoretical Amount of Heat.	Real.	Average.
6	53	3.830	23.5	2.30	2,997	2,907	90	...
7	58	3.424	25	2.30	2,997	2,844	153	...
8	57	3.554	23	2.32	3,023	2,901	122	...
9	59	3.424	24.5	2.315	3,016	2,893	123	...
10	58	3.427	23	2.28	2,971	2,846	125	...
11	57	3.542	23	2.32	3,023	2,891	132	...

End-Result.

	Charge-Tests.	Blank-Tests.	Difference, Charge to Blank-Tests.	Amount of Heat Required for Decomposition of 0.2141 g. $CaCO_3$.	Heat Evolved in Slag-Formation.
	G-Cal.	G-Cal.	G-Cal.	G-Cal.	G-Cal.
6	90	65	25	96	+ 121
7	153	87	66	96	+ 162
8	122	82	40	96	+ 136
9	123	66	57	96	+ 153
10	125	101	24	96	+ 120
11	132	79	53	96	+ 149
Average,					+ 140

Heat of Formation of Ferro-Calcic Singulo-Silicate (*FeO*, 40.30 : *CaO*, 28.00; *SiO*$_2$, 31.70 *Per Cent.*).—In order to furnish 0.28 g. of CaO, there was required 0.4996 g. of $CaCO_3$: its decomposition into CaO and CO_2 absorbs 224 g-cal. In Table V., the result for test No 14 is too low on account of the charge not having been completely fused. Tests Nos. 12, 13, and 15 give concordant results. The heat of formation for 1 g. of the mixture is therefore 193 g-cal.

TABLE V.—*Heat of Formation of Ferro-Calcic Silicate* (*FeO*, 40.30: *CaO*, 28.00; *SiO*$_2$, 31.70 *Per Cent.*).

(Total water-equivalent = 1,303 g.; time = 60 sec.; weight of charge = 1 g.)

Blank-Tests.

Test No		Current.	Difference in Potential	Room-Temperature	Corrected Rise in Temperature.	Actual Amount of Heat.	Theoretical Amount of Heat	Difference, Actual and Theoretical Amounts of Heat. Real.	Average.
		Amperes.	Volts.	Deg C	Deg. C	G-Cal	G-Cal	G-Cal	G-Cal.
12	a.	54	4.009	24.5	2.424	3,158	3,100	58	70
	b.	54	4.020	24.5	2.45	3,192	3,109	83	
13	a.	54	4.061	26.0	2.515	3,277	3,141	136	116
	b.	54	4.053	27.0	2.479	3,230	3,134	96	
14	a.	52	4.049	24.0	2.350	3,062	3,015	47	63
	b.	52	4.070	24.0	2.386	3,109	3,031	78	
15	a.	55	4.090	25.5	2.570	3,349	3,222	127	120
	b.	55	4.091	25.5	2.565	3,342	3,228	114	

Charge-Tests.

Test No	Current.	Difference in Potential	Room-Temperature	Corrected Rise in Temperature.	Actual Amount of Heat.	Theoretical Amount of Heat	Real.	Average.
12	54	3.833	25	2.305	3,003	2,965	38	...
13	54	3.885	26	2.355	3,068	3,005	63	...
14	52	3.880	24.5	2.200	2,867	2,889	— 22	...
15	55	3.854	25.5	2.415	3,147	3,036	111	...

End-Result.

	Charge-Tests.	Blank-Tests.	Difference, Charge to Blank-Tests.	Amount of Heat Required to Decompose 0.4996 g. $CaCO_3$.	Heat Evolved in Slag-Formation.
	G-Cal.	G-Cal.	G-Cal.	G-Cal.	G-Cal.
12	38	70	—32	224	+ 192
13	63	116	—53	224	+ 171
14	— 22	63	—85	224	+ 139[a]
15	111	120	— 9	224	+ 215
Average,					+ 193

[a] Omitted.

The amounts of heat evolved in the formation of these three singulo-silicates indicate an increase with an increment in the percentage of lime. On account of the fact that many difficulties were encountered which caused a considerable number of failures, the gaps between 0 and 12 per cent. of CaO, and between 12 and 28 per cent. of CaO, were not filled. In order to carry an experiment of this character with greater speed, it seems necessary to construct an apparatus of larger capacity, taking a charge of perhaps 5 g., and to have a regulator for keeping automatically a constant current. Some work carried on recently in a Mahler bomb suggests a modification of the Le Chatelier method of determining the heat developed in slag-formation—namely, to inclose the slag-charge in a platinum capsule, imbed this in charcoal, and proceed in the usual manner.

We wish to express our obligation to the Laboratory of Heat Measurement at the Institute of Technology, and to Prof. C. L. Norton for the interest he has taken in this research.

TABLE VI.—*Thermo-Chemical Data for the Heats of Formation of Silicates.*

Starting From	G-Cal. Developed per G-Mol.	G-Cal. Developed per 1 g. Silicate Formed.	Starting From.	G-Cal. Developed per G-Mol.	G-Cal. Developed per 1 g. Silicate Formed.	Reference.
(FeO, SiO_2)......	10,600	80	(Fe, Si, O_3)........	254,600	1,929	Le Chatelier, Compt. Rend., 1895, cxx., 623.
(MnO, SiO_2)	5,400	41	(Mn, Si, O_3)......	276,300	2,109	Le Chatelier, Compt. Rend., 1896, cxxii., 80.
(BaO, SiO_2)...........	14,700	69	(Ba, Si, O_3)......	328,100	1,540	Tschernobaeff, Revue de Métallurgie, 1905, ii., 729; Electrochemical and Metallurgical Industry, 1906, iv., 72.
(CaO, SiO_2)............	17,850	154	(Ca, Si, O_3).......	329,350	2,839	
($2\ CaO, SiO_2$)........	28,300	165	(Ca_2, Si, O_4).....	471,300	2,740	
($3\ CaO, SiO_2$)...........	28,550	125	($Ca_3, Si\ O_5$)......	603,050	2,645	
(SrO, SiO_2)...............	17,900	110	(Sr, Si, O_3).......	329,100	2,019	
($Al_2O_3, 2\ SiO_2$)..........	14,900	67	(Al_2, Si_2, O_7).....	767,500	3,457	
($3\ CaO, Al_2O_3, 2\ SiO_2$)	33,500	86	(Ca_3, Al_2, Si_2O_{10})	1195,550	3,065	
($2\ H_2O, Al_2O_3, 2\ SiO_2$)	43,800	170	(H_4, Al_2, Si_2, O_9)	927,420	3,595	
(Li_2O, SiO_2)	65,100	720	(Li_2, Si, O_3).........	347,100	3,856	
(Na_2O, SiO_2)...........	45,200	370	(Na_2, Si, O_3)......	326,100	2,673	
(CaO, Al_2O_3)...........	450	3	(Ca, Al_2, O_4)......	524,550	3,220	
($2\ CaO, Al_2O_3$)..........	3,300	15	(Ca_2, Al_2, O_5).....	658,900	3,079	
($3\ CaO, Al_2O_3$).........	2,950	11	(Ca_3, Al_2, O_6)......	789,050	2,922	
(SiO_2 35.5, FeO 39.7, MnO 1.0, CaO 11.4, MgO 2 7, Al_2O_3, 9.2, Cu 0.42, S 0.42)......	....	133	....	....	....	Richards, Electro-chem. and Met. Ind., 1907, v., 266.
($2\ FeO, SiO_2$)........... (FeO 70 80, SiO_2 29.20)	22,236	109	(Fe_2, Si, O_4)......	333,636	1,637	Hofman-Wen.
(FeO 57.58, CaO 12.00, SiO_2 30.42)........	....	140	....	....	....	Hofman-Wen.
(FeO 40 30, CaO 28.00, SiO_2 31.70)	....	193	....	....	....	Hofman-Wen.

Application of Descriptive Geometry to Mining-Problems.

BY JOSEPH W. ROE,* NEW HAVEN, CONN.

(Pittsburg Meeting, March, 1910.)

MANY questions arising in the work of the mining engineer may be solved quickly and with sufficient accuracy by the methods of descriptive geometry; but, unfortunately, this subject is more often considered from a mathematical view-point than as an effective tool for solving practical problems. Some of the principles involved are merged into the ordinary operations of mechanical drawing, and are used unconsciously by the engineer in his every-day work. The rest are stowed away on the book-shelf with his forgotten or rarely-used college text-books. One reason for this condition of "innocuous desuetude" may lie in the unfortunate and often cumbersome notation which it has been found necessary to introduce. If some of these principles can be recalled to the engineer's attention in terms of every-day use they may prove of practical bearing.

Recently, in talking with several mining engineers, I used some of the drawing-room methods of a machine-designer, to the evident surprise of those around me. These men were perfectly familiar with the methods, but their application to mining-work had not occurred to them. At their suggestion the present paper is offered, not in any way as a treatise, but merely to suggest a certain point of view, in hope that it may lead others far more familiar with mining-problems to pursue the subject further.

I. To Find the Strike.

As the simplest and most fundamental problem, let it be required to find the strike of a vein when the location and elevation of three points on the outcrop are known. It will be assumed for the present that the vein is a plane.

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* Sheffield Scientific School, Yale University.

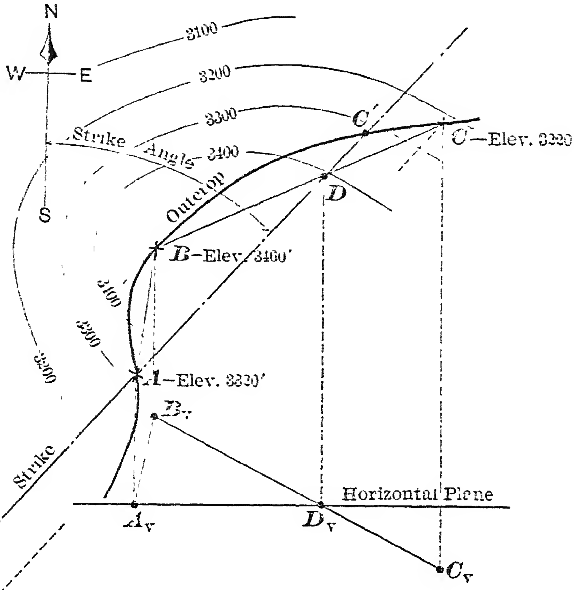


Fig. 1.

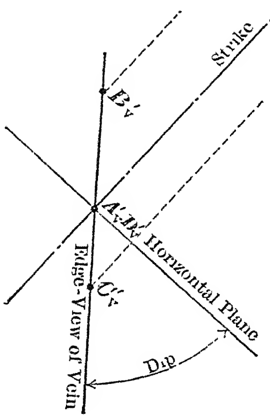


Fig. 2.

In Fig. 1 let A, B, and C be three points on an outcrop, their elevations, as found by observation or shown by contour-levels, being as noted. Connect A and B, B and C, giving the plan-view of two lines lying in the plane of the vein. If a side-view be drawn, as shown beneath the plan, A, B, and C will appear as A_V, B_V, and C_V (the v-subscript indicating a vertical projection), and a horizontal plane through A will appear as a horizontal line through A_V. D_V, where the line B_V C_V cuts the plane, is the side-view of a point in the vein on the same level as A. Projecting up to the line BC, this point appears in the plan-view at D. Two points in the plan-view, such as A and D, on the same level and both in the vein, determine the strike. A saving in line-work might be made by drawing the

line representing the side-view of the horizontal plane through A itself instead of through its projection, A_V, using the same construction otherwise. It is clearer, for the present purpose, to show the side-views separately instead of partly superimposed.

If a point on the outcrop at the same level as A can be located, as C'', the line from A to C'' will at once give the strike. A third point, B, will, however, be necessary to determine the dip, as a plane cannot be located with less than three points.

II. To Find the Dip.

Having found the strike, take a side-view of the vein looking up the strike-line. The points A, B, C, and D and the plan-view will appear as shown. The line through B'_V, C'_V, etc., Fig. 2, is the edge-view or end-projection of the vein, and the dip is, of course, the angle included between this and the horizontal plane.

The dip might also be found by passing a vertical projecting-plane through B in Fig. 1 perpendicular to the line of strike, and then revolving the line of intersection of this plane and the vein into the plane of the paper. This is the characteristic method of descriptive geometry, but when the engineer wants to detail some part which stands at odd angle with the rest of his drawing, as, for instance, a hand-hole plate or a sky-light, he does not adhere rigidly to his right-angled projections, but draws an oblique view looking squarely at the detail, showing it directly and giving the dimensions without any troublesome revolvings. By taking a side-view up the line of strike many problems which otherwise might be troublesome are reduced to the very simplest terms. As examples of this take the following:

III. Depth of a Shaft.

Given the points on an outcrop as before, how deep must a shaft be sunk at some point, S, to strike the vein?

The strike and dip having been found, we have the side-elevation of the vein, as shown in Fig. 3. The point, S, from which the shaft is to be sunk appears as S_V at its proper elevation, and the depth of the shaft is obtained at once by scaling S_V E_V.

IV. Length of a Tunnel.

Suppose it is proposed to drive a tunnel with, say, a 5-per cent. up-grade from some point, T, Fig. 4, to the vein. Where will it hit the vein, how long will it be, and what will be its bearings?

Find the strike and side-view. Project the point T to T_V. Draw a line, $T_V\ M$, parallel to the horizontal plane, and $T_V\ N$,

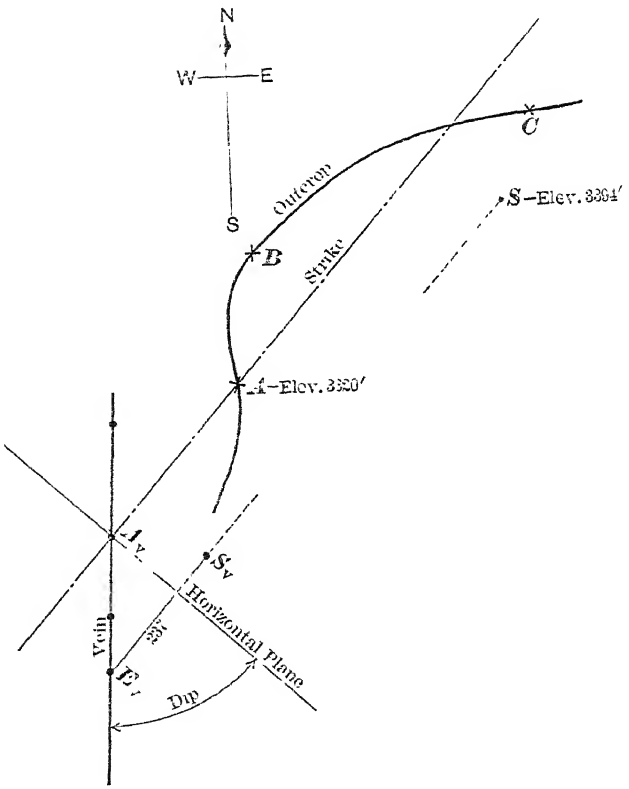


Fig. 3.

making an angle with $T_V\ M$ corresponding to the grade. $T_V\ F_V$ is, to scale, the length required. In the plan-view it is obvious that the tunnel will appear at right angles to the strike, which gives at once the bearing-angle. Project F_V up to meet this line, as at F, and we have the plan-view of the point of intersection of the tunnel and the vein. The elevation of F may be obtained by trigonometry, or more simply still by

scaling the difference in level between F_v and the horizontal plane through A_v.

V. Location of a Tunnel.

Let A, Fig. 5, be the mouth of an inclined shaft and 1, 2, and 3 be drifts from it, both shaft and drifts following the vein. B is a vertical prospecting-shaft of a certain depth on an adjoining property. What will be the direction and

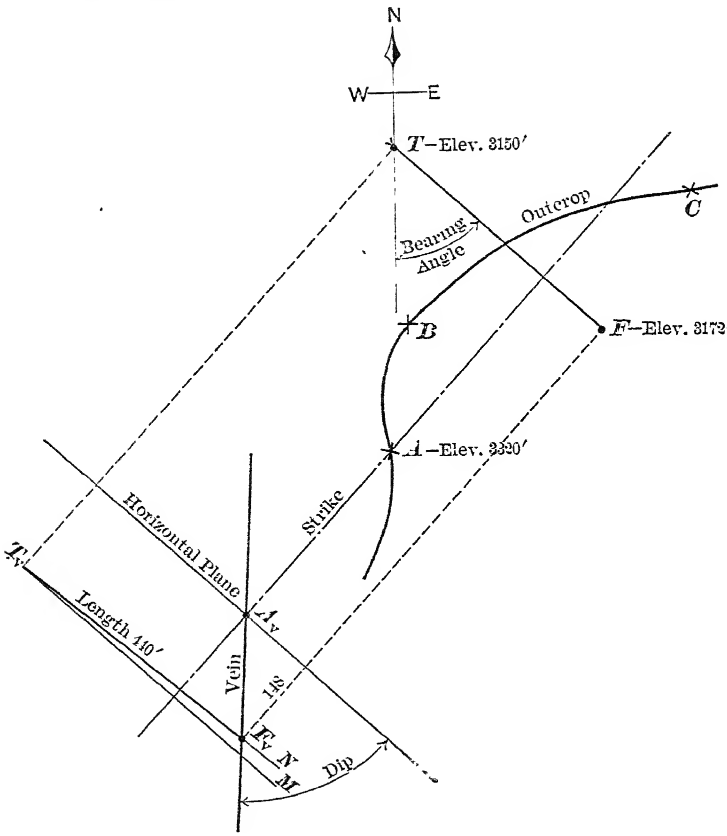


Fig. 4.

length of the shortest drift from the bottom of B to pick up the vein and still keep on the "B" side of the line?

The drifts 1, 2, and 3 are presumably horizontal, and since they lie in the vein they are parallel to the strike. In the end-view they will appear as 1_v, 2_v, 3_v, and will determine the end-view of the vein, the vertical shaft, B, appearing at B_v B'_v. A horizontal line from B'_v will cut the vein at C_v. The line

XY through C_V, parallel to the levels, is the plan-view of the intersection of the vein and a horizontal plane through B_V, and any horizontal drift from B'_V must intersect the vein somewhere along this line. In the plan-view the shortest drift would be $B\ C'$. But $B\ C'$ would run out of the property. BC therefore would be the limiting position of the drift to have it entirely on the "B" side of the line.

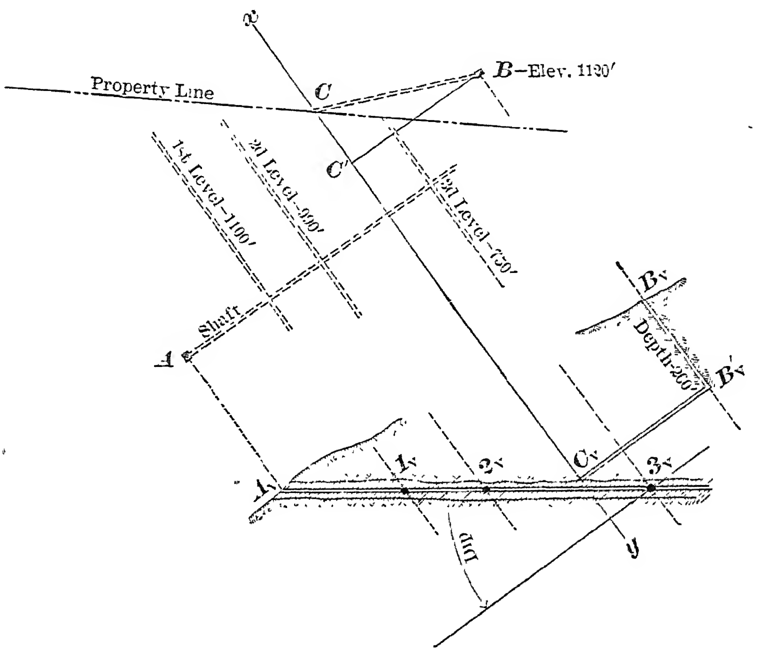


FIG. 5.

VI. To Find the Bearings of a Shoot.

Suppose we wish to find the bearings of a shoot in the vein ABC of Figs. 1 to 4. Let the shoot appear on the outcrop at B, Fig. 6, and have a pitch of 19° down towards the NE.

A side-view of the point B will show B_V 140 ft. above the level of A. Any line through B with a pitch of 19° from the horizontal must cut the level of A at a distance of mn from the foot of the perpendicular through B_V. All points in the vein on the same level as A lie somewhere along the strike-line. With B as a center and mn as a radius swing the arc XX, cutting the strike-line at some point, S. Connecting B and S, we

have the bearing of the shoot. In the case shown we would expect the shoot to reappear on the outcrop at *T*. This construction might be used to locate a possible reappearance of a shoot in rough country overgrown with underbrush.

A somewhat similar construction would give the bearing of a drift, lying in an inclined vein and itself inclined at some grade angle, say 10 per cent. Lay off to scale in an end-view a parallel 10 ft. above a horizontal through A_V. Project D_V, the intersection of the parallel with the vein, to intersect an arc, *yy*, of 100 ft. radius swung from *A*, as at *D*. *AD* is the bearing desired.

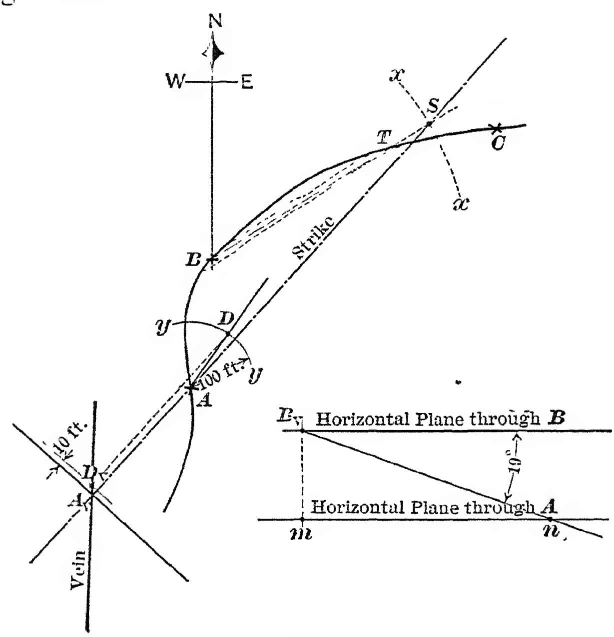


FIG. 6.

VII. TO FIND THE THICKNESS OF A VEIN.

Given the outcrop and dip of the hanging- and foot-wall, how thick is the vein and what is the strike?

Case 1.—Let the side-wall be vertical. Let the outcrops be as shown in the side-view of Fig. 7, their intersections with the ground or other horizontal plane being A_V and B_V. In the plan the side-wall appears as a line. Project A_V and B_V to *A* and *B*. Draw any line, *yy*, cutting the walls at C_V and D_V. Through C_V and D_V draw $C_V n$ and $D_V n_1$, making the given

dip angle with the horizontal. The perpendicular distance between these lines is the thickness of the vein. The strike would, of course, be perpendicular to the direction in which the

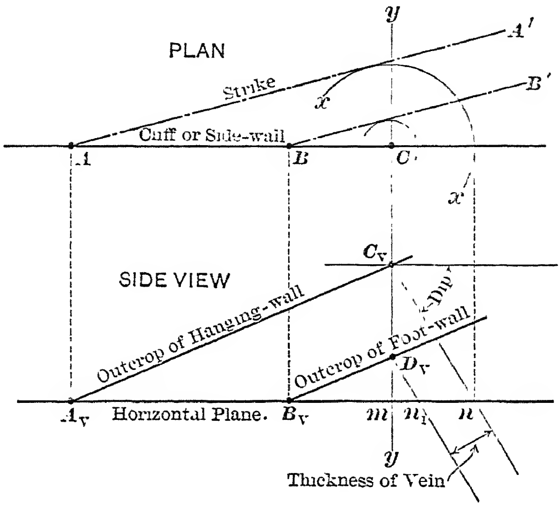


FIG. 7.

dip is measured. A construction giving it more accurately would be as follows: Any plane through C_V having the given

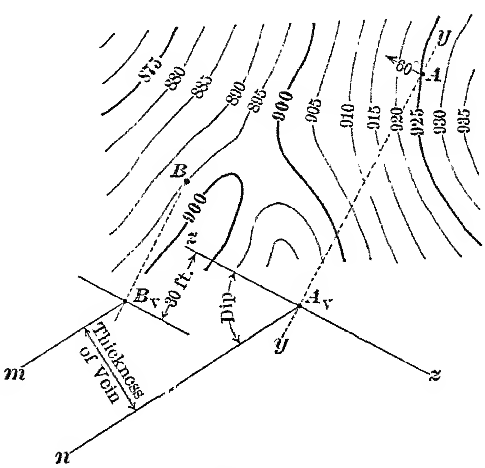


FIG. 7A.

dip angle must cut the horizontal plane at a distance mn from m. With C as a center and mn as a radius swing the arc XX.

The strike-line, AA', passes through A tangent to XX. BB' is parallel to AA', or it may be found in the same manner as AA'.

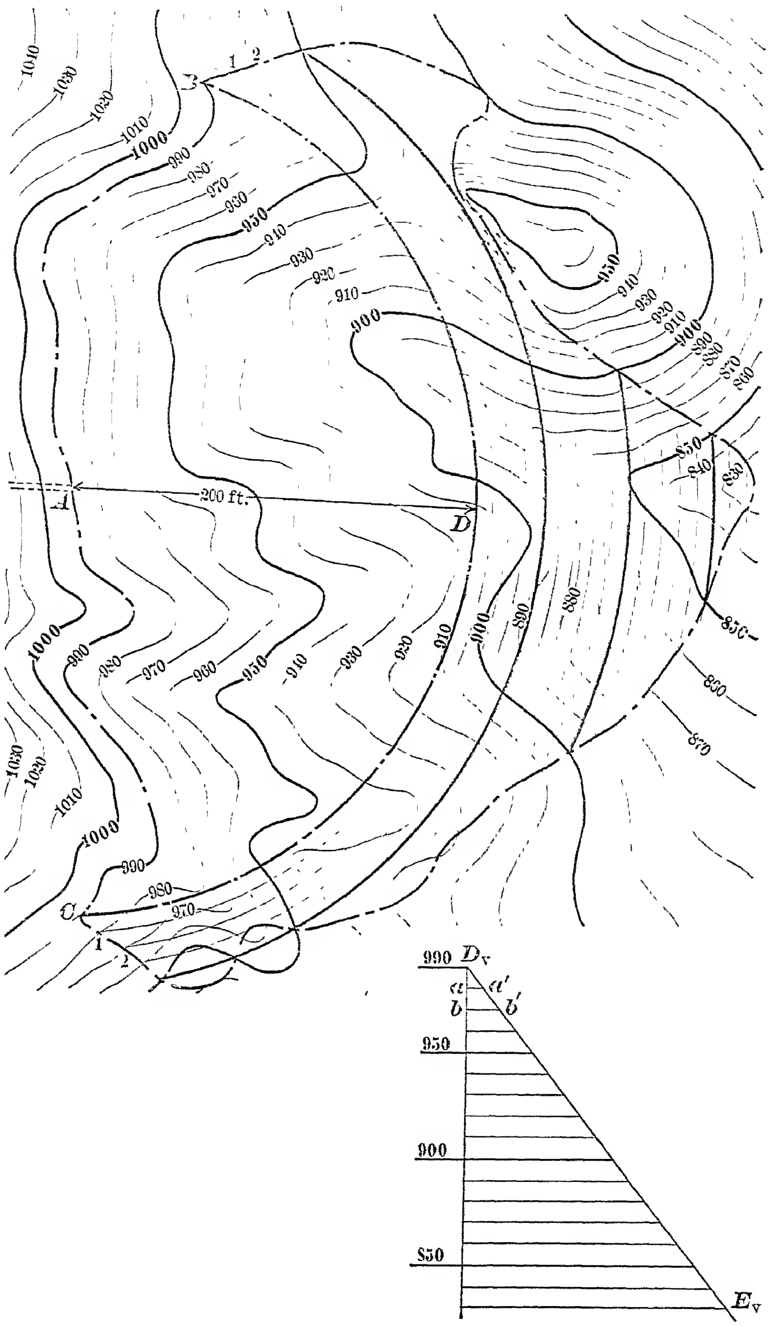
Case 2.—Let the surface be irregular, as, for instance, in Fig. 7A. Let the location and elevation of A and B and the angle and direction of dip be as shown. Draw a projection-line, yy, perpendicular to the direction of dip. Draw any line, zz, perpendicular to yy. Letting A_V be the side-view of A, lay off the dip angle as shown. Lay off B_V on a projecting-line parallel to yy at a distance of 30 ft. below zz. Through B_V draw $B_V\,m$ parallel to $A_V\,n$. The perpendicular distance between these lines is the thickness of the vein.

VIII. To Find the Limits of an Ore-Dump.

Given the location of a shaft or tunnel, A, Fig. 8, on a certain contour-level as shown, to find the outer limits of a dump formed by filling in on a level with A for a certain radius, say 200 ft.

With A as a center and $AD = 200$ ft. swing an arc, BC, cutting the contour-level of A at B and C. The area between this arc and the contour through A is, of course, the top of the dump. Between B and C the side of the dump will extend down hill a varying distance, depending on the angle of rest of the material and the contour of the ground. Draw an auxiliary view showing a vertical section of the face of the dump. The sloping side appears as the indefinite line, $D_V\,E_V$, at an angle equal to that at which the material "hangs up," say 40°. Draw a series of levels, aa', bb', etc., corresponding to the contour-lines of the map, and to the same scale.

The side of the dump is a portion of a cone, with a vertical line through A as its axis. The top of the dump and the various levels, being parallel planes perpendicular to this axis, cut the cone in a series of concentric circles. All points on the surface of the ground at an elevation of 980 ft. lie on the 980-ft. contour-line, and all points on the face of the dump at 980 ft. elevation lie on an arc concentric with BC, and having a radius aa' greater than that of the top. Wherever this arc meets the 980-ft. contour, as at 1, 1, there is a point on both the face of the dump and the surface of the ground. The remaining points, as 2, 2, etc., may be located by swinging the arcs from A or by measuring the increase in the radius, as bb', etc., out from



A
D
C
200 ft.
1
2
3
1000
1010
1020
1030
990
980
970
960
950
940
930
920
910
900
890
880
870
860
850
840
830
D_v
a
b
a'
b'
990
950
900
850
E_v

FIG. 8.

the arc BC, to cut the proper contour-line. Connecting the series of points so located, we have the completed outline of the dump.

There may be places where the edge does not cross a contour for some distance. Any uncertainty as to its location may be cleared up by interpolating on the ground and on the face of the dump, as shown on the shoulder of the knoll.

With the limits located and the surface-contours on the face drawn, calculation of the tonnage becomes a mere matter of mensuration. The area of the dump at each level is readily obtained from the contours with a planimeter. Each area will be bounded by a circular arc along the face of the dump and the corresponding irregular contour-line where the dump is in contact with the ground. The volume of each layer may be found by the end-area method or by the prismoidal formula.

IX. Ore-Dump on a Property-Line.

In practice the problem might easily occur in the reverse form. Given a point, A, Fig. 9, from which it is desired to fill, and a property-line, XXX. How much tonnage-capacity is there available in filling out to the line and extending, say, 200 ft., as before, wherever the property-line is not encountered? In the previous case the upper edge of the dump was regular and the outer one irregular. In this, the limits of a portion of the outer edge are set by the property-line, and the upper edge will, in consequence, be irregular.

As before, we have the crest of the dump starting out from B and C, Fig. 9, the two points on the contour through A at the given limiting distance of 200 ft. For a short distance the crest will be circular, with its center at A, and the outer edge will be found as before. It is evident, however, that the crest cannot swing out far before the property-line is encountered. To determine the side of the dump where it is affected by the side-line we may proceed as follows: Draw an auxiliary side-view. The level of the top, 990 ft., appears as shown, also the side of the dump and the parallels corresponding to the contour-levels of the map. For any points on the property-line, such as a and a', which are on the 890-ft. level, it is evident that a vertical projection of the crest cannot come within the distance mn of a or a' without having the dump over-run the

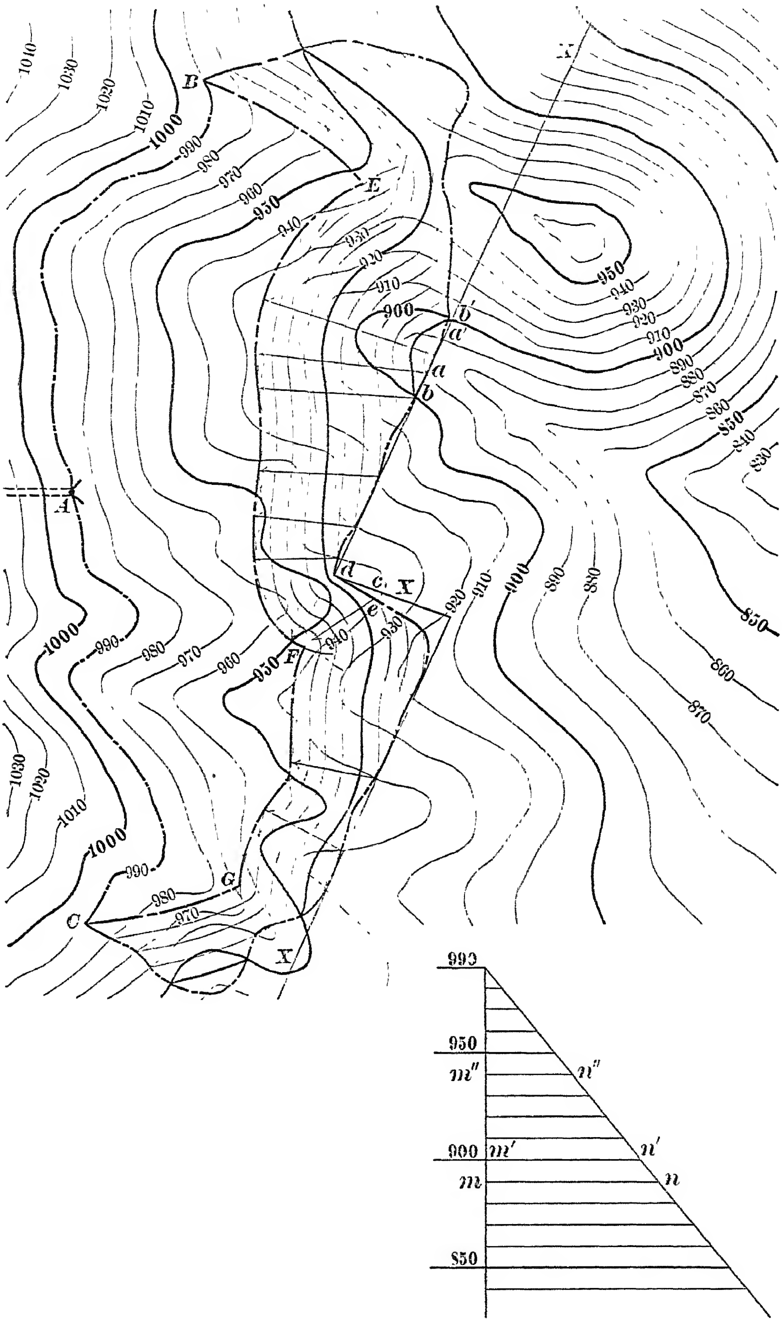


FIG. 9.

side-line. From a and a', then, swing arcs with mn as a radius. These arcs limit the position of the crest so far as a and a' are concerned. Similarly, for b and b' on the 900-ft. level, arcs swung with $m'n'$ as a radius locate the crest so far as b and b' are concerned. Continuing this process for the intersection of each contour-line and the property-line, we obtain a series of arcs. The innermost arc, or the one towards the uphill side, limits the crest. Drawing a fair line tangent to the inside arcs, we have the crest EFG. An arc not used, such as that from c, indicates that as far as c is concerned the crest might come out to that arc; but the position of that from d, further in, shows that before the material could be filled out to the outer arc it would be over-running at d.

While the line EFG is the farthest limit to which the crest can be carried, it does not follow that the foot of the dump will run out to the property-line at all points. For instance, with the material falling away at an equal angle on all sides, it is evident that F cannot be pushed out far enough to cover the extreme corner without over-running on both sides. A portion of the corner and also any high points, such as those opposite E and G, will remain uncovered. The outline of the base is obtained, as in Fig. 8, by measuring out a distance from the crest equal to that from the vertical in the auxiliary view to the intersection of the sloping side of the given level. Thus, to locate the point e on the 940-ft. contour, cut that contour with an arc swung from F with a radius equal to $m''n''$, etc.

The contour-levels on the side of the dump corresponding to those of the ground will be a series of equidistant lines, the distance between them being equal to the difference between the successive lines $m'n'$, mn, etc., in the auxiliary figure. Wherever the dump-contour and the ground-contour for the same level intersect there is a point on the outer edge or foot of the dump.

X. Location of Ridge for Maximum Capacity.

If it is desired to store a maximum tonnage on a given space the ore must rise from the boundary-line on all sides at the angle of rest of the material. If the ground is level the sides will meet in a point for a square or round area, in a straight horizontal ridge for a rectangular area, and in a straight, in-

clined ridge for an area bounded by non-parallel straight sides. When, however, the ground is broken and irregular, as is generally the case, the sloping sides will no longer be plane surfaces, and will meet in a sloping and irregular crest. It is an advantage to locate this crest in advance, for it is only when the material is deposited along this line that the maximum storage-capacity of the ground is utilized. This crest, then, would indicate the proper location for a dumping-trestle, for material can be filled in along this line until it begins to over-run on both sides at the same time.

Let it be desired to cover the tract shown in Fig. 10, from some point, such as F, on the upper side. Locate F and the course of a dumping-trestle from which the entire tract may be filled in. Use the auxiliary side-view as before and work in from the boundary-lines, the first radii being mn, as at A, B, C, the second $m'n'$, etc. Here every fifth line is used, and the intermediate ones are filled in only along the crest where they are needed. The construction used will be clear to one who has followed the previous two problems. A contour on the dump, starting from C, would pass B at a distance of mn away, and would pass A at a distance of $m'n'$ away, etc. The dump-contours located from each side will be found to intersect along a broken line, FHG, which is the crest. This line is the theoretical course for a trestle and may be adhered to as closely as is deemed desirable. A slight spur would have to be run to the left from a little below H as far as I to fill in the left-hand corner completely, but the capacity gained would hardly justify the expense.

XI. To Plot the Outcrop of a Vein, which May be Considered as a Plane Surface.

Descriptive geometry methods may be used to determine the probable outcrop of a vein. First, let the vein be free from faults and foldings, and the strike and dip be known for some definite position, as A in Fig. 11. Draw an end-view looking up the strike-line. For convenience, let this end-view be drawn with the line XX through A perpendicular to the strike as the horizontal plane through A. Let the series of lines parallel to XX be the end-views of the horizontal planes corresponding to the contour-lines of the map. If a number of

parallel planes be cut by a common secant plane, the resulting intersections form a series of parallel lines. The lines aa', bb', etc., projected up from the intersection of the vein and the various levels, are in reality the contour-levels for the vein,

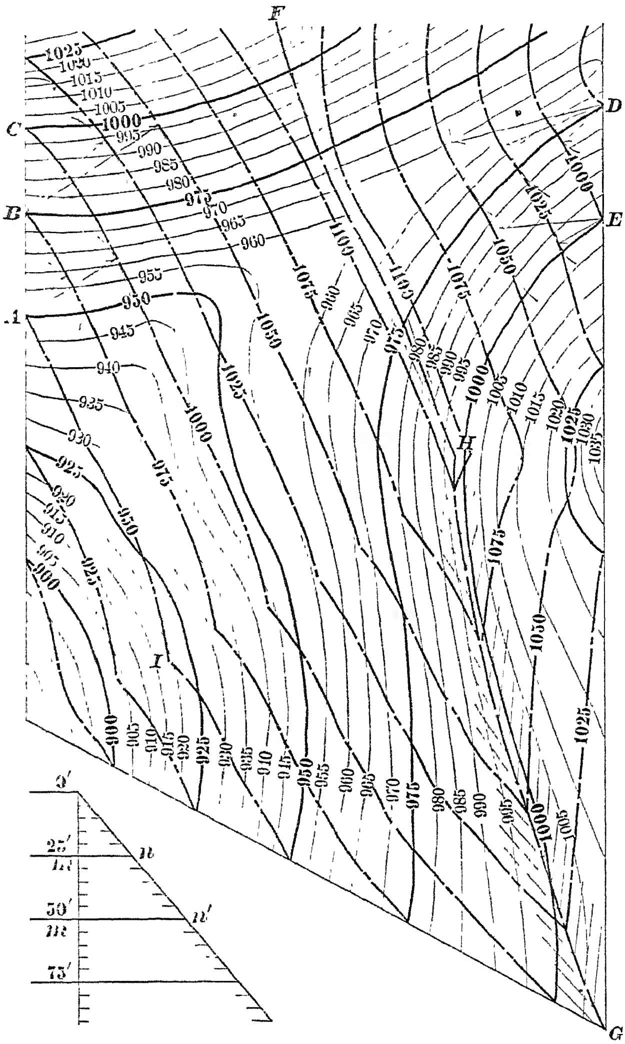


FIG. 10.

which in this case become a series of parallel lines. Wherever, therefore, the line AA' and the 710-ft. surface-contour cross, we have a point in the vein and on the surface of the ground; that is, on the outcrop. Similarly, the intersections of the

line *aa'* and the 700-ft. contour give points on the outcrop. By locating these intersections for each level and connecting them up we have the outcrop. This method can be used only for comparatively small areas where the vein may be considered as a plane surface. When the vein is faulted to any great extent the treatment becomes complicated, so much so as to preclude its consideration in this paper. When the vein is continuous but folded, as, for instance, a coal-seam, a method of determining the outcrop may be used, which is illustrated in Figs. 12 to 12B.

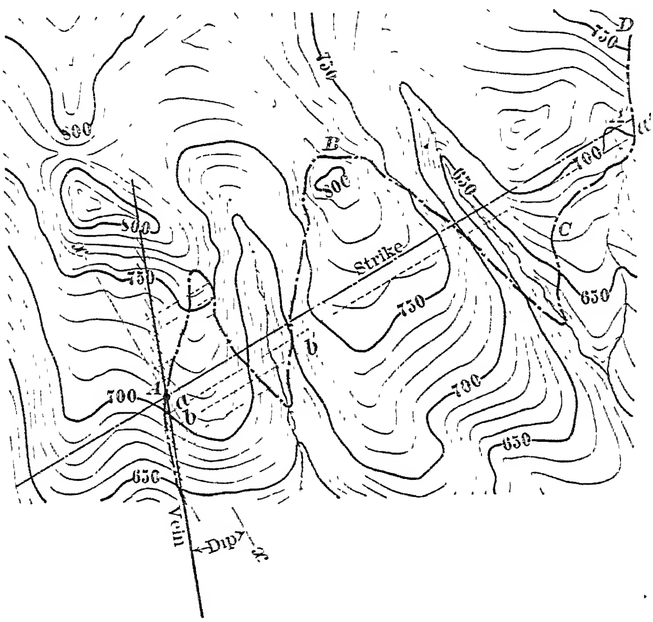


FIG. 11.

XII. To Determine the Outcrop of a Folded Vein or Seam.

Let Fig. 12 be a contour-map of an area on which an irregular outcrop has been partly located, as, for instance, at the stations marked with an *X*. At various other points, as 6, 9, 11, 19, drill-holes have been sunk and the seam located at the depths noted. The form of the folded seam may be determined if we can establish its contour-lines corresponding to those on the surface, and where these seam-contours intersect

those of the surface of equal elevation we have points on the outcrop.

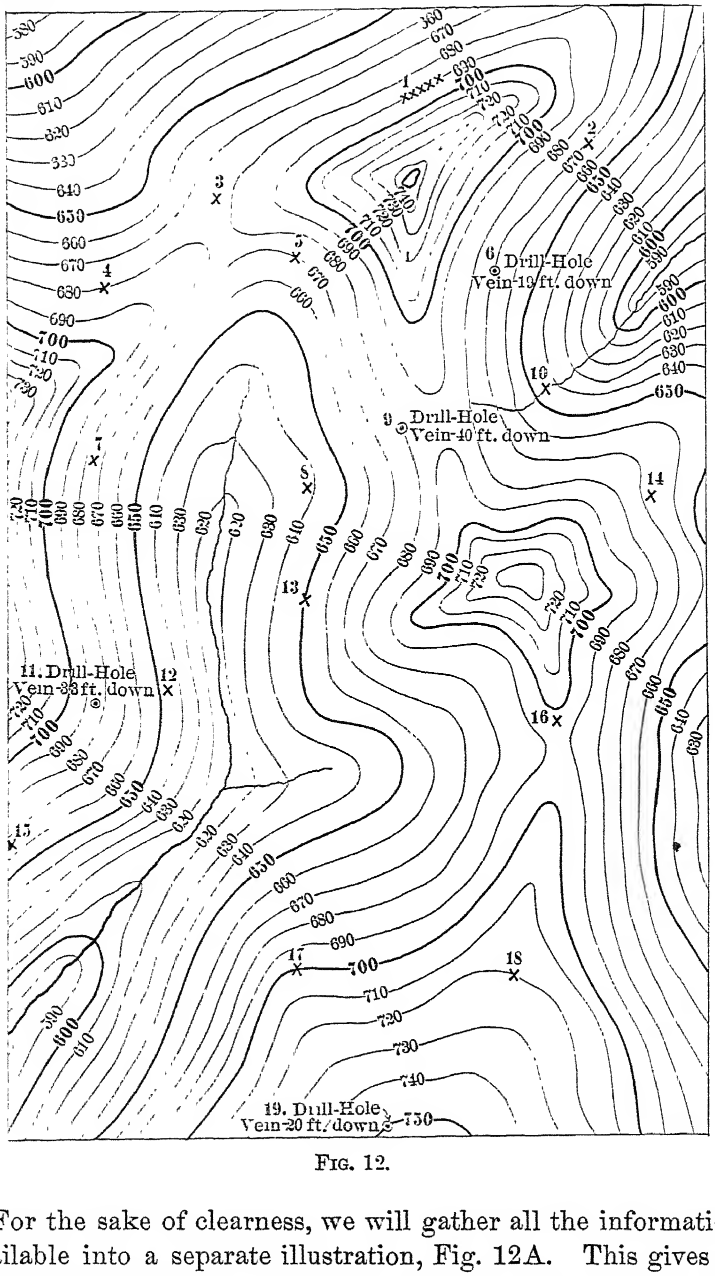


FIG. 12.

For the sake of clearness, we will gather all the information available into a separate illustration, Fig. 12A. This gives us stations 1 to 19, with the elevations of the surface of the seam

for each, as noted. A general inspection shows that there is a syncline near the middle of the tract, the depression running

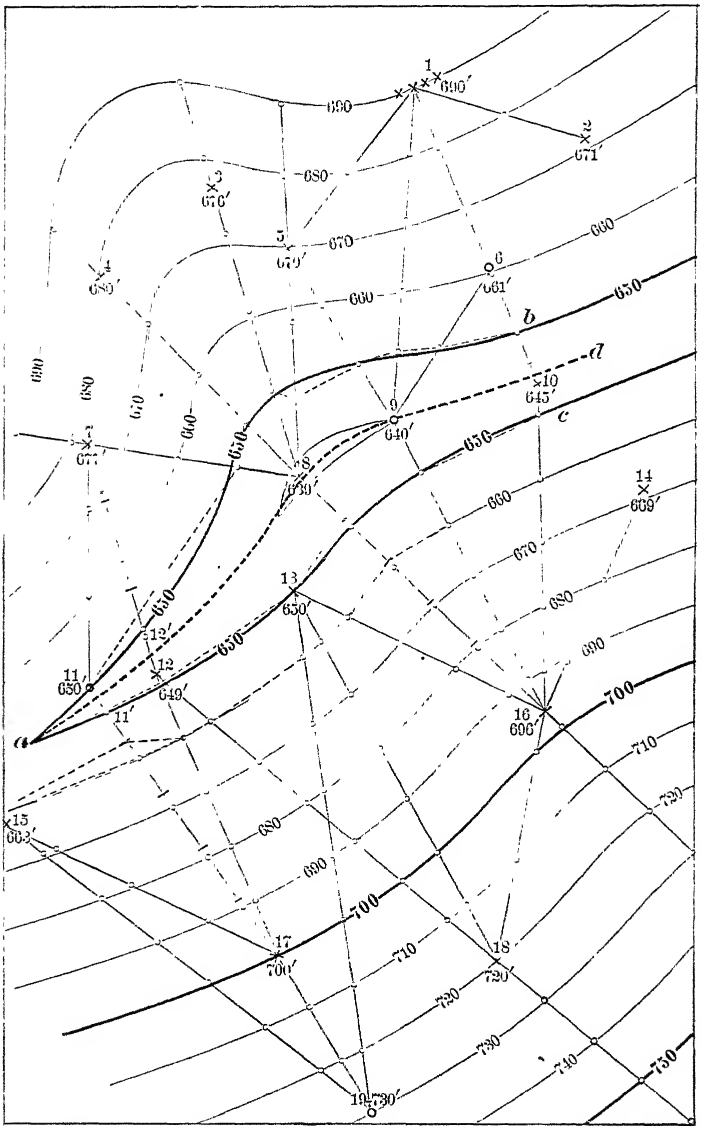


FIG. 12A.

NE–SW., since stations 8, 9, 10, 11, 12, 13 are all at nearly the same level. Assuming, for a start, that the dip from the

higher points, as 16, 17, 18, etc., towards the depression is substantially constant, lines may be drawn connecting the stations,

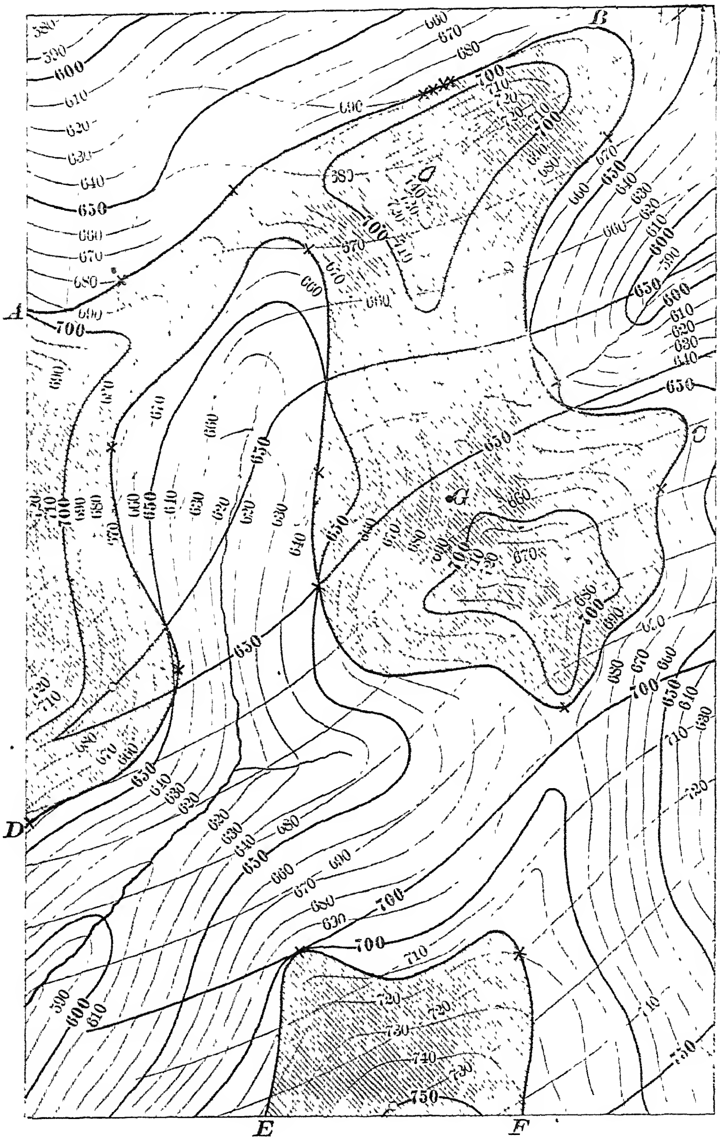


Fig. 12B.

and the elevations at the contour intervals may be interpolated, as on lines 13–16 and 13–18. Where necessary the levels may

be extrapolated, as on the extensions of lines 8–16 and 12–18, but points so located are, of course, not so reliable as the interpolations. They aid, however, in obtaining the general direction of the contours, and as the seam has been eroded at these points great accuracy is not needed.

In this way a series of points of equal elevation are obtained for both sides of the depression. The auxiliary lines, such as 12–18 and 9–16, which are substantially perpendicular to the depression, will be steeper, cross more contours, and give more reliable interpolations than lines more nearly parallel. In interpolating with these lines care should be used that they connect stations on the same side of the depression, as, for instance, 11–17. Station 12 is probably on the same side as 17, but 11 may be, and in fact is, on the other side. This may be determined as follows: The points on the 650- and 660-ft. levels line up well except those marked with a cross-bar on lines 11–17 and 7–12. Rejecting these two sets and connecting the others, we have the broken lines *ab* and *ac*, which give the general location of two 650-ft. contours. Assuming that the trough lies midway between these lines, we obtain the dotted line *ad* as its location. If now we take a new point, 11′, on the opposite side of *ad* and at the same distance from it, and interpolate the line 11′–17, we obtain points agreeing well with the others. In like manner, a new set of elevations may be obtained for the lines 7–12, 8–16, and 6–10.

Connecting the various points of equal elevation by curves, we obtain seam-contours, as shown in Fig. 12A. If these seam-contours of Fig. 12A be superimposed on the surface-contours of Fig. 12, we will have a series of points where contours of equal elevation intersect, which will determine the irregular outcrops, *ABCD* and *EF* of Fig. 12B. The shaded portions indicate the areas under which the seam lies. Elsewhere it has weathered away. The probable depth of the seam at any point may be obtained by subtracting the elevation of the seam from that of the surface, as indicated by the respective contours for that point. For instance, at *G* the depth is 690 ft. less 655 ft., or 35 feet.

The value of this contour-method, where the outcrop is overgrown or obscured by soil, is obvious. An excellent example of its application will be found in the Masontown Folio of the

U. S. Geological Survey, where structure-contours have been drawn for the upper surface of the Pittsburg coal-seam, showing its elevation above sea-level. Over large portions of this area the coal has been eroded. Where observations were obtainable a stratum was observed to maintain a substantially constant interval below. This lower stratum was but little eroded, and was used to obtain the theoretical position of the eroded portion of the coal-seam. A number of hills, high enough to pierce the seam thus located, were found to contain portions where its presence would not otherwise have been suspected. This indirect method of using parallel strata was also utilized where the seam lay too deep for observation direct, in this case the stratum being above the seam.

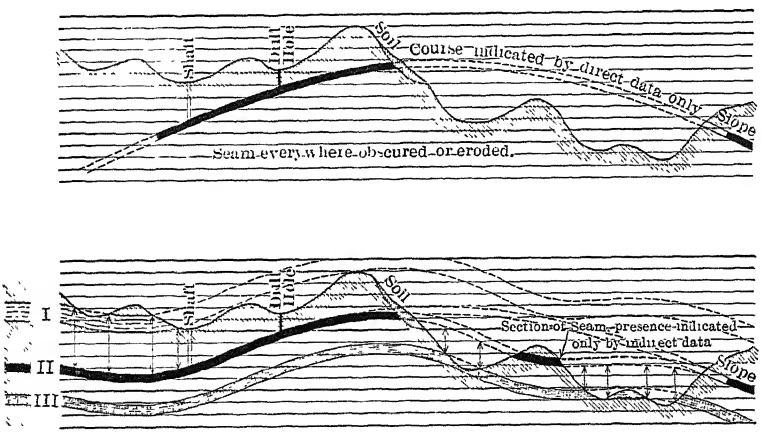


FIG. 13.

Fig. 13 illustrates how parallel strata may be used to corroborate and supplement direct data. This use of parallel strata has been described by M. R. Campbell[1].

The direct data of the upper section, Fig. 13, give only three points on the seam, which is everywhere obscured or eroded. If the intervals between I. and II. and II. and III. in the columnar section be established, and if I. and III. be located at various outcrops, as shown in the lower section, then the intervals may be set off from these outcroppings, and will greatly increase our information on the surface to be contoured.

[1] *Trans.*, **xxvi.**, 298 (1896).

I wish to acknowledge my indebtedness to Prof. John D. Irving, who was engaged on the work of the Masontown quadrangle, and who suggested this structural contour-method as one of the useful applications of descriptive geometry to mining-work.

A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest.

BY J. B. DILWORTH, PHILADELPHIA, PA.

(Pittsburg Meeting, March, 1910.)

In estimating the investment-value of a mining-property or plant, the value of which decreases with operation, it is often necessary to know the sum which must be set aside periodically from earnings either to renew the plant at the end of its life, or to return the capital involved when the earning-power of the investment has ceased. In such cases it is manifestly incorrect simply to divide the total amount to be retired by the number of periods (usually years) during which the investment shall be active, as the sums periodically withdrawn have a certain interest-earning power varying with the length of time they are held and with the opportunities afforded for their investment.

But though the interest may be considered to accumulate regularly year by year, yet the rules and tables for compound interest do not apply, owing to the addition of a fixed amount of new money to the total at the beginning of each period.

Briefly stated, the problem of sinking-funds is, to find the amount of money which must be periodically set aside at a certain rate of interest, regularly compounded, to yield a certain sum in a given length of time. In most practical cases the interest is compounded and the fresh amount added at the same time—usually once a year—and on this basis the problem may be solved as follows:

Let $S =$ total amount to be retired,
Let $r =$ interest rate : $R = 1 + r$,
Let $n =$ life of the sinking-fund or the amortization-period,
Let $x =$ the sum set aside at the end of each year.

Then the amount of the sinking-fund at the end of first year is x, which at the end of n years will have increased by compound interest to xR^{n-1}. Similarly, another amount, x, will be set aside at the end of second year, which will in turn increase

to xR^{n-2} at the end of the amortization-period; and so on to the last payment at end of the nth year, when the amount will be simply x. Therefore, the total amount, S, at the end of the amortization-period, will be

$$S = xR^{n-1} + xR^{n-2} + xR^{n-3} \quad . \quad . \quad . \quad . \quad + xR^3 + xR^2 + xR + x.$$

This equation is reducible to simpler form by multiplying through by R and subtracting the first equation from the result, thus:

$$\begin{array}{rlll} RS = & xR^n + xR^{n-1} + xR^{n-2} & . \; . \; . \; . & + xR^4 + xR^3 + xR^2 + xR \\ S = & \qquad xR^{n-1} + xR^{n-2} & . \; . \; . \; . & + xR^4 + xR^3 + xR^2 + xR + x \\ \hline RS - S = & xR^n & & -x, \text{ or} \end{array}$$

$$S\,(R-1) = x\,(R^n - 1), \text{ or } x^* = S\frac{R-1}{R^n-1} = S\frac{r}{(1+r)^n - 1}.$$

As an illustration of the method of using this formula, let it be assumed that the amount to be retired, S, is \$70.95; the interest rate, r, 5 per cent., and the amortization-period, n, 16 years. Substituting in the formula, $x = 70.95\,\dfrac{0.05}{(1.05)^{16} - 1}$ $= 3$, *i.e.*, \$3 set aside at the end of each year at 5 per cent. interest, annually compounded, would amount in 16 years to \$70.95.

Results obtained with the above formula may be checked by finding by actual multiplication and addition what \$1 set aside each year will amount to in the given time at the assumed rate of interest. Then a simple proportion will give the sum which must be annually invested at the same interest rate to furnish the required sum at the end of the amortization-period. In the foregoing example it may be found arithmetically that a \$1 a year fund, interest 5 per cent., will amount in 16 years to \$23.65. Now if \$23.65 is produced by a \$1 a year fund, \$70.95 will be produced by a \$3 fund, or, expressed in the form of a proportion, 23.65 : 1 : : 70.95 : 3.

Table I. gives the amount which a \$1 a year fund will afford in from 1 to 50 years with interest-rates varying from 2 to 8 per cent. By the method of proportion just illustrated these

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* If payments are to be made and interest compounded semi-annually then

$$\tfrac{1}{2}x = S\frac{\frac{1}{2}r}{(1+\frac{1}{2}r)^{2n} - 1}, \text{ or } x = S\frac{r}{(1+\frac{1}{2}r)^{2n} - 1}.$$

results may be applied to any case in hand. No originality is claimed in presenting this table—more extensive ones may be found in the files of savings banks and insurance companies—but it is given here in the belief that it may serve some to whom the more complete ones are not readily accessible.

TABLE I.—*Sinking-Fund Table.*

Time: At End of Year.	*Rate of Interest.* 2 Per Cent.	3 Per Cent.	4 Per Cent.	5 Per Cent.	6 Per Cent.	7 Per Cent.	8 Per Cent.
1st	1.00	1.00	1.00	1.00	1.00	1.00	1.00
2d	2.02	2.03	2.04	2.05	2.06	2.07	2.08
3d	3.06	3.09	3.12	3.15	3.18	3.21	3.25
4th	4 12	4 18	4.25	4.31	4.37	4.44	4.51
5th	5.20	5.31	5.42	5.52	5.64	5.75	5.87
6th	6.31	6.47	6.63	6.80	6.98	7.15	7.34
7th	7.43	7.66	7.90	8.14	8.39	8.65	8.92
8th	8.58	8.89	9.21	9.55	9.90	10.26	10.64
9th	9.75	10.16	10.58	11.03	11.49	11.98	12.49
10th	10.95	11.46	12.01	12.57	13.18	13.82	14.49
11th	12.17	12.81	13.49	14.21	14.97	15.78	16.65
12th	13.41	14.19	15.03	15.91	16.87	17.89	18.98
13th	14.68	15.62	16.63	17.71	18.88	20.14	21.50
14th	15.97	17.09	18.29	19.60	21.01	22.55	24.22
15th	17.29	18.60	20.02	21.58	23.27	25.13	27.15
16th	18.64	20.16	21.82	23.65	25.67	27.89	30.33
17th	20.01	21.76	23.70	25.84	28.21	30.84	33.75
18th	21.41	23.42	25.66	28.13	30.90	34.00	37.45
19th	22.84	25.12	27.68	30.54	33.76	37.38	41.45
20th	24.30	26.87	29.79	33.06	36.78	41.00	45.76
21st	25.78	28.68	31.98	35.72	39.99	44.86	50.43
22d	27.30	30.54	34.26	38.50	43.39	49.01	55.46
23d	28.84	32.46	36.63	41.43	46.99	53.44	60.90
24th	30.42	34.43	39.10	44.50	50.81	58.18	66.77
25th	32.03	36.46	41.66	47.72	54.86	63.25	73.11
26th	33.67	38.56	44.33	51.11	59.15	68.68	79.96
27th	35.34	40.71	47.10	54.66	63.70	74.48	87.35
28th	37.05	42.93	49.98	58.39	68.52	80.70	95.34
29th	38.79	45.22	52.98	62 31	73.64	87.35	103.97
30th	40.57	47.58	56.10	66.43	79.05	94.46	113.29
31st	42.38	50 01	59.34	70.75	84.80	102.07	123.35
32d	44.23	52.51	62.72	75.29	90.88	110.22	134.22
33d	46.11	55.08	66.23	80.05	97.34	118.93	145.96
34th	48.03	57.73	69.88	85.05	104.18	128.26.	158.63
35th	50.00	60.46	73.67	90.31	111.43	138.24	172.32
36th	51.99	63.28	77.62	95.82	119.11	148.91	187.11
37th	54.03	66.18	81.72	101.61	127.26	160.34	203.08
38th	56.11	69.16	85.99	107.69	135.90	172.56	220.33
39th	58.24	72.24	90.43	114.08	145.05	185.64	238.95
40th	60.40	75.40	95.05	120.78	154.75	199.63	259:07
41st	62.61	78.67	99.85	127.82	165.04	214.61	280.79
42d	64.86	82.03	104.84	135.21	175.94	230.63	304.26
43d	67.16	85.49	110.04	142.97	187.50	247.78	329.60
44th	69.50	89.05	115.44	151.12	199.75	266.12	356.97
45th	71.89	92.72	121.06	159.68	212.73	285.75	386.52
46th	74.33	96.51	126.90	168.66	226.50	306.75	418.44
47th	76.82	100.40	132.98	178.10	241.09	329.22	452.92
48th	79.35	104.41	139.30	188.00	256.55	353.27	490.15
49th	81.94	108.55	145.87	198.40	272.94	379.00	530.37
50th	84.58	112.80	152.70	209.32	290.32	406.54	573.80

The Giroux Shaft at Kimberly, Nev.

BY C. EVERARD ARNOLD, KIMBERLY, NEV.

(Pittsburg Meeting, March, 1910.)

THE Giroux Consolidated Mines Co. is equipping a five-compartment shaft at Kimberly, Nev., which will serve the Alpha mine.

The depth of this shaft, January, 1910, is 963 ft., and of this depth 948 ft. was opened by raising; 15 ft. of sinking was done so that the bearers, collar-set, and three shaft-sets might be installed and concreted, and with this done it was possible to erect the necessary surface-equipment for resuming shaft-timbering on the completion of the raise. The arrangement of the shaft-timbering is shown in Fig. 1.

At the 1,000-ft. level of the Alpha, and 750 ft. SW. from the Alpha shaft, a 14- by 18-ft. station was cut, and a 25-h-p. air-hoist was installed. From here the raising commenced on July 5, 1909. The raise was cut approximately 13 by 21 ft. in area, and supported by timbering with square sets, illustrated in Fig. 2, which shows the division into four compartments. The end compartments, which were used as chutes, had the center girts heavier than in the others on account of the wear from falling rock. The chute-lining was braced by inserting 8- by 8-in. stiffeners parallel to and half way between successive caps. Similarly, the lagging on the wall sides of the cage-way and the compartment for pipes and ladder was stiffened with 4- by 6-in. timbers. In the pipe-and-ladder compartment the chute-stiffeners were reinforced with 8- by 8-in. struts, and as the cage-way was partitioned off by braced lining, there was practically no chance of rock entering the ladder-way in case a chute burst into the cage-way.

Over the two center compartments the last two sets were always bulkheaded with 8- by 12-in. timbers, shown in Fig. 3, and from the lower bulkhead the cage sheave-wheel was suspended. After a round of holes had been blasted and the loose

ground picked down, the average time required to hoist, plumb, and block a set of timbers was 2 hr., although the operation was occasionally performed in 1 hr. 30 minutes.

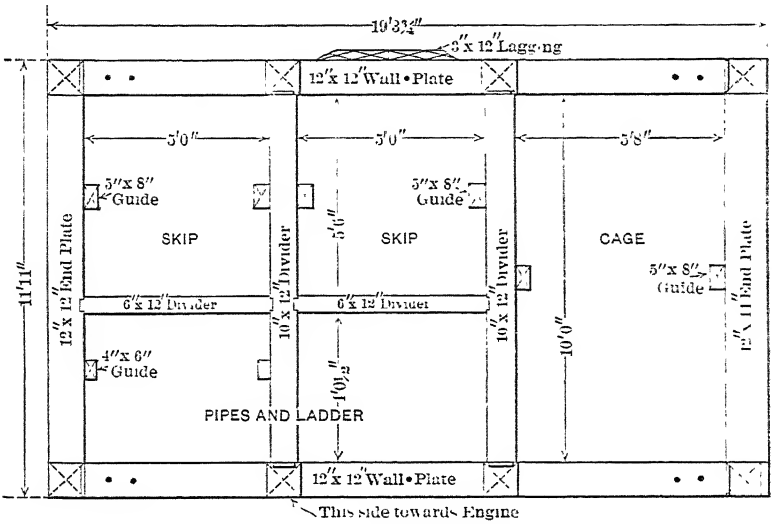

FIG. 1.—PLAN OF SHAFT TIMBERING.

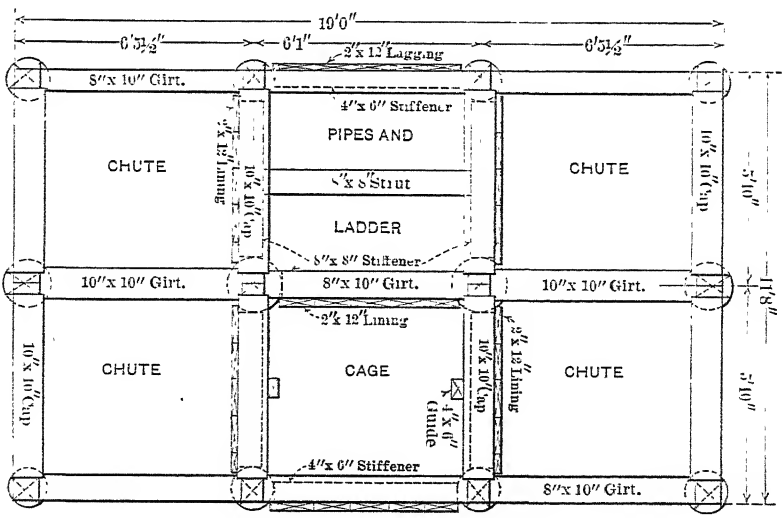

FIG. 2.—PLAN OF RAISE TIMBERING.

When possible, the chutes were kept almost full in order to minimize the chances of injury in case men should happen to fall. For the purpose of protecting the lower timbers of the

chute, breakers or platforms of 10- by 10-in. timber were spiked on to the girts, in the manner shown in Fig. 4, at intervals of seven sets. On these breakers the rock piled up and formed its

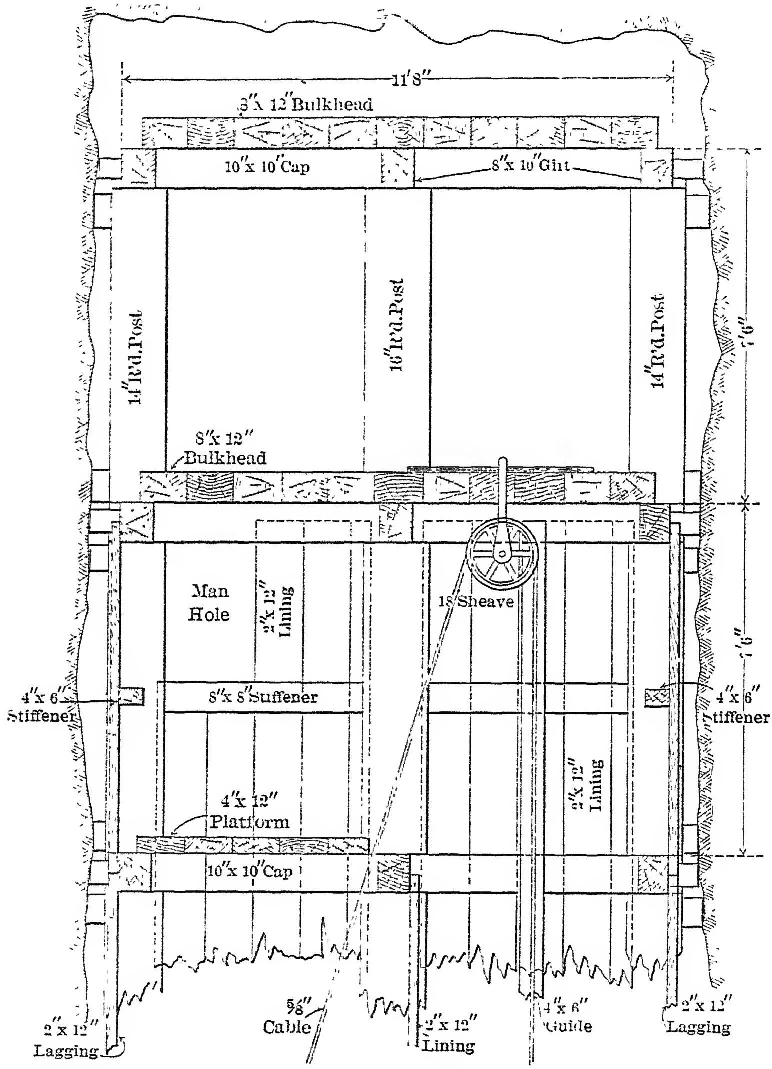

Fig. 3.—Part Section through Cage-Way and Pipe- and Ladder-Compartment of Raise.

own slope. Trouble was often caused by large rocks dropping into the chute, and on reaching the gate these rocks would have to be drilled and blasted before they could pass through.

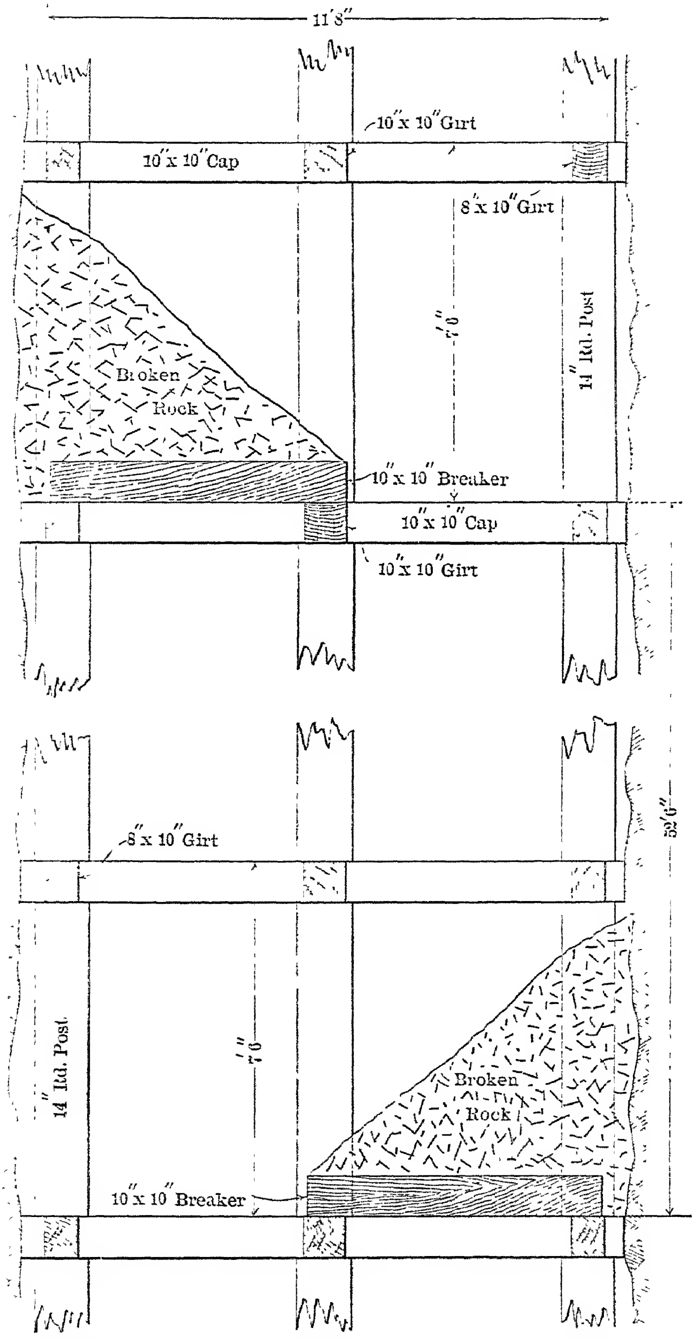

FIG. 4.—DIAGRAM OF SCHEME USED FOR BREAKING FALL OF ROCK IN CHUTE.

The drilling was done entirely with Waugh stopers, No. 8 C, and these machines proved remarkably satisfactory. Three were used for raising the entire distance, one being kept in reserve in case of accident to the others, although all three machines were used on the last 75 ft. of the raise. Each machine wore out one chuck and one tappet, and beyond replacing these, no repairs were required.

The shaft is wholly in limestone, and for the greater part the ground was favorable for fast raising. Normally, on blasting a round of 36 holes 7 ft. deep there would be room for a set of timbers and the back would support itself well. Sometimes the ground was so shattered by intersecting slips that only half the round could be blasted with safety. This procedure made room for half the set, which was then blocked before the remaining holes were blasted and the set completed. At one or two spots the ground encountered could be removed with a pick, and in these cases each post was placed as soon as there was room for it. At 125 ft. from the surface there was a small patch of ground so hard that two rounds of 40 holes were blasted before a set could be placed.

Excellent ventilation was maintained by using two Roots pressure-blowers, size 1, running in tandem. One was placed at the foot of the raise, drawing the foul air through an 8-in. air-pipe and delivering through an 8-in. air-pipe and a 6-in. pump-column to the other blower, which was above ground.

After the 1,000-ft. and 770-ft. levels of the Alpha were connected, the raise was served entirely from the latter level. Three shifts were worked continuously, and 180 days were required to complete the raise, the rate of advance being 5.27 ft. per day, or 158.0 ft. per month.

The progress per month was: July, 167; August, 150; September, 133; October, 143; November, 165; December, 190; total, 948 ft.

The total cost of the raise was $67,508.89, which is at the rate of $71.21 per foot, distributed as follows:

		Cost Per Foot.		Cost Per Foot.
Labor:	Raising,	$16.69	Air-compressor,	$10.00
	Tramming,	5.09	Supplies,	17.24
	Timber-framing,	1.76	Hoisting,	15.15
Blacksmithing,		1.19	Proportion of general underground expense,	3.44
Machine-shop,		0.11		
Teaming,		0.54	Total,	$71.21

As the development of the mine proceeds, 75 per cent. of the timber used in this raise will be available for other purposes; hence the raise will eventually be credited with 75 per cent. of the cost of that timber; $10 per foot of raise represents the timber-cost, so that the true cost of the raise is $71.21, less $7.50, or $63.71 per foot.

The work was performed under the direction of William Hogan, mine superintendent, formerly of the West Stewart mine, Butte, Mont.

I wish here to express publicly my cordial thanks to Frank P. Mills, general manager of the Giroux Consolidated Mines Co., for his courtesy in furnishing the necessary data given in this paper.

Professional Ethics.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910.)

In June, 1906, I delivered to the graduating class of Lehigh University an address upon this subject, the substance of which, with sundry omissions and additions, was subsequently repeated, in October of the same year, in an extempore address before the Schenectady branch of the American Institute of Electrical Engineers. The latter, as stenographically reported, was published in the *Proceedings* of that society for November, 1906, and was referred to by President John Hays Hammond in his address at the Chattanooga meeting of October, 1908. Requests have been received from various quarters for the republication of this address by the Institute. This I do not regard as necessary or proper; but I yield to that demand so far as to reproduce in the *Transactions* a summary of my views, as expressed in the two addresses above mentioned.

I considered the subject of professional ethics under the several heads of the positions likely to be occupied by engineers—namely, as authors, employees, agents, or private or public advisers. Under these heads I said in substance:

1. *Authors.*—The author is party to an implied contract with his reader, in which he covenants that he has something to say, and will say it clearly, honestly, and fully. He ought, therefore, to avoid all obscurities or ambiguities of style; "mixed figures" of rhetoric (which are essentially insincere, because they present as seen by him pictures which could not possibly have been sanely seen by anybody—as when Wendell Phillips said the time would come when Liberty would "stand by every new-born child, and drop in its cradle the school-house and the ballot-box!"); misuse of quotation-marks, by making them include anything else than *verbatim* quotations; second-hand references to authorities not actually verified by the author, without specific statement of the fact, etc. And a technical author ought, in good faith, to furnish his reader with all the aids to further detailed inquiry which he himself possesses. In other words, he is bound to give, not only his own conclusions, but also such a statement of his grounds and authorities as will fairly show the range of his own knowledge.

2. *Employees.*—Under this head, both duty and policy dictate a loyal devotion to the interest of the employer and a consideration of duties rather than rights. Even from a selfish stand-point, loyalty commands to-day the highest price in the market.

3. *Agents.*—The obligations of this relation are as clearly defined by law as by personal honor. No agent may have secret personal relations, affecting the interests of those for whom he is openly acting. The payment of commissions to purchasing agents is a case in point. Sometimes, by reason of trade-agreements, a manufacturer cannot sell to a customer below a certain price, yet is permitted to pay the agent who procures the customer for him. If that agent is at the same time the agent of the purchaser, he cannot honorably receive such a commission without the knowledge of his principal. But, if the rules of trade make this the only way of securing a reduction of price, he may accept such a commission and turn it over to his employer, in which case he should insist upon receiving it

in a check to his order, and indorse the said check to the order of his employer, so that there may be no doubt in any quarter as to the disposition which he has made of it.

4. *Private Advisers.*—Under this head, perhaps the most frequent function is the writing of letters of recommendation. Do not give general letters of this kind if you can avoid it. A letter which the candidate carries in his own pocket, and of which the contents are known to him, has little weight the first time it is exhibited, and less and less weight as time goes on; and it becomes dog-eared and dirty through frequent presentation or pocket-wear. Moreover, a letter of personal recommendation is really a piece of advice to an unknown client; and it cannot possibly be made as clear and effective as it ought to be, if it is drawn in a general form, without knowledge of the particular requirements of the said unknown client. Even for the benefit of your best friend, it is much better to give a letter, authorizing him to use your name as a reference, and promising to answer, confidentially and fully, any inquiry that may be made of you concerning him. Such an inquiry will come only from some party seriously considering the engagement of the candidate. When it does come, it can be answered with intelligence and freedom; and if the man is, in your opinion, a good man for the place, he will be ten times as likely to get it, upon such a statement from you, as upon the strength of your vague general recommendation, presented by himself. But in all such cases it should be remembered that the inquirer, not the candidate, is your client; and, even in giving the most favorable opinion, you should carefully state the extent of the knowledge upon which it is based. You have no right to try to get a place for a friend by misleading or incomplete advice to your client, his prospective employer.

Under the head of private advice falls the advice given to clients as to investments. Here honor requires, in the first place, that the client should not be deceived as to your ability to deal with his particular problem. The numerous instances in which scientific experts in one line have undertaken, with disastrous results, to give opinions in another line, furnish sufficient illustration of this proposition. There was a time when experts were scarce in this country, and a chemist was regarded as competent to pronounce judgment upon a piece of mining-

ground. But the situation has been so changed that, nowadays, even a trained mining engineer cannot be trusted for such a judgment unless, in addition to his general knowledge, he has also a special acquaintance with the nature and history of the particular kind of mining and the particular district concerned. One of the most eminent mining experts within my acquaintance never undertakes to report upon an enterprise in a region not familiar to him without engaging, at his own expense, a local expert of his own selection, to post him as to such matters as his own examination might miss. (I note that Mr. Hammond approves this course.)

I can recall instances of really able and unquestionably honest mining experts who, lacking such local aid, and ignorant of local conditions and history, have reported hundreds of thousands of tons of ore, on the strength of scattered "float" from a comparatively small outcrop.

It is true, that a thoroughly trained mining engineer should be able, by keen and faithful study, to get at all the essential facts for himself; but he ought to be frank with his client beforehand, and let the latter clearly understand the situation.

Private advice is also dependent upon the nature of the question presented. The client may say, "Here is the report of the owner of the property. I have agreed to buy it, if his statement be true; and I employ you to verify that statement." Such a commission is entirely honorable; yet the discharge of it is not altogether agreeable or safe for him who performs it. For the condition stated is narrow, and its honorable fulfillment is liable to serious subsequent misunderstanding. If, for instance, the expert should find that the representations of the vendor, which he was sent to verify, were true, but that other and equally promising mining-property in the neighborhood could be bought for a lower price, he could not properly include in his report any reference to that more profitable alternative. Yet his favorable report as to the vendor's statement submitted to him, might be construed afterwards as a recommendation of the purchase, which was not really submitted to him at all. This situation occurs, perhaps, most frequently under the English system of organization of mining companies, according to which a company is organized on the basis of a vendor's statement, for the purpose of purchasing a certain

mining-property, provided that statement be indorsed as correct by a competent expert, selected to represent the company. Upon the report of that expert, the existence of the company itself, and all the profits of its promotors, depend; and it would be, of course, unprofessional for him to declare that, while the representations of the vendor were true, he had heard of another investment, which he thought was better!

For the mining expert himself, it is highly desirable to insist, if practicable, upon a wider authority, permitting him, not only to verify the vendor's report, but also to advise as to the purchase itself. Either duty, I think, may be honorably undertaken: but in the former case, the expert should take care that his task and responsibility are clearly expressed, especially if his report is likely to be published afterwards, and thus converted from private to public advice.

5. *Public Advisers*.—Under this head fall the expert reports which are published in the prospectuses of mining companies, and the testimony of experts in court. Under the first head, I have only to say: (1) That an expert has the right to insist that his report should not be garbled, by fragmentary publication or otherwise, so as to support a scheme which it did not contemplate or approve; and (2) that his protest against such misuse of his statements should be made promptly. As I said to the Electrical Engineers, "Don't come limping in, after a scheme has failed, to protest that you never meant to approve it in all respects. Speak quick, or shut up!"

Among public advisers are those who appear as expert witnesses in court. The difference between an expert witness and an ordinary witness is, that the former is expected, while the latter is forbidden, to give his opinions, and to state, among the grounds of those opinions, the facts and conclusions reported by others, as well as those personally verified by himself. The expert's position is, that he has been called, as *amicus curiæ* —a disinterested "friend of the Court"—to assist judge and jury in the interpretation of the evidence, a part of which may be his own testimony as to facts observed by himself. But as to his statement, whether of such facts or of his opinions, the expert should not forget that he has sworn to state the truth fully and only. It has often been urged that experts should be employed by the Court, and not by either party. This plausible

notion involves the selection of such expert advisers without knowledge of their theoretical views. I have had some experience of the practical working of that plan; and I can testify that it results in opinions, usually voluminous and vague, which need further expert interpretation before they can become useful guides to judge or jury. Moreover, the Court may innocently select an expert who is a partisan of a particular scientific theory. On the whole, therefore, I am thoroughly convinced that our present American system, under which expert witnesses are presented by each party, to be cross-examined by the other party, is, with all its inconveniences and costs, best adapted to elucidate the truth and promote the ends of justice. It must be confessed that, under that system, the expert witness becomes an advocate; for neither party in a law-suit would produce such a witness without knowing beforehand that his opinions were favorable to its contention. Yet it still remains true that the expert witness is, after all, a sworn adviser of the Court. Under these circumstances, what is his professional duty?

I have answered this question by declaring:

a. That no expert can honorably serve in such a capacity, without having previously made sure, for himself and his clients, that he can honestly defend their contention.

b. That if, after having, on a preliminary retainer, studied the case, he cannot do this, he should decline to appear as a witness in court, either for that side or the other.

c. That his preliminary study of the case should be so thorough as to warrant him in propounding and defending in court his expert theory. He may fairly appear as a partisan of that theory, though not of the party aided by it. And, in defending his view, he is still under oath, and must not shrink from the candid acknowledgment of new evidence presented by the other side, not previously known to him, or deal with such evidence evasively or controversially, for the purpose of protecting his own reputation. He is justified in saying that he has made up his deliberate opinion upon all the facts observable or ascertainable by him, and cannot change it upon the presentation of new evidence which he has had no opportunity to test and weigh.

Above all, the expert witness should not try to be smart or

witty under cross-examination. In such an encounter his hands are tied; for he is under oath, while the inquiries and insinuations of his antagonist are unrestrained.

The best expert witness I ever heard courteously assumed that the cross-examining attorney was sincerely in quest of information, and gave it to him with fraternal and pedagogical cordiality which simply paralyzed his attack! I have seen many a witness beat a lawyer in repartee, and lose the case for his client; and I have seen many an honest witness confused and embarrassed under cross-examination—but the jury saw it also, and gave the verdict to his side!

It frequently happens that an expert is required to answer categorically, "Yes" or "No," a question which cannot be thus answered without giving place to unjust misconstruction. In such a situation, the expert witness should not "fence" or "dodge" (thereby giving the impression that he is afraid of the inquiry), but simply give the categorical answer required, and then ask the privilege of further explanation. This the Court will always grant, to his complete protection and the discomfiture of the ingenious cross-examiner.

It happens sometimes, especially in mining law-suits, that an expert who has expressed in court a certain opinion is forced by later developments to change it. For instance, he may have testified to the separate occurrence of two lodes, while deeper mining has afterwards shown that these two, uniting, have become one—with lamentable and disastrous results, perhaps, to conflicting claims of ownership under the U. S. laws. Now, such changes of opinion upon later evidence are very common and perfectly honorable. And an expert who has once honestly advocated in court a given opinion has a perfect right to retract that opinion afterwards. But if he be wise, he will not announce that change as an expert under fee, in another lawsuit. Let him state it in a technical essay or lecture, if he thinks it worth publishing—but not before a jury, to which it would have to be explained in connection with the unpleasant circumstance that both the erroneous first opinion and its later correction have been paid for. Moreover, an expert offering such an explanation in court will be forced either to confess that he came to a wrong conclusion the first time, or else to assert that his opinion was correct then, but has been changed

by later evidence—in other words, that he might change it again upon further evidence. Such declarations will inevitably have an unfavorable immediate effect upon the case of his client, and a permanent effect upon his own reputation. This does not apply with equal force to a changed opinion which has been already publicly declared elsewhere. Yet, even in that case, it is not advisable to appear in court as an expert contradicting one's former testimony. To give voluntary expert testimony against one's former clients is, of course, dishonorable. But when this consideration is not involved, an unnecessary recantation, under oath and fee, is still inadvisable. In view of my former statement that the expert witness is a partisan, not for a side, but for an honest opinion, it might fairly be said that his opinion is really his client, and that if it has been overthrown he had better not become the partisan of another opinion before a judge and jury.

6. *The Advantage of Frank Publicity.*—I concluded my address to the Electrical Engineers with the following paragraphs, which I take the liberty of quoting here, since I have no desire to modify them:

In conclusion, is there not some handy rule which would help us in every case of doubt? The Golden Rule and the Ten Commandments we all want to keep; but they cannot always be applied in haste or with certainty. Let me suggest something practical, which has religion in it, but not enough to hurt or scare you. "*Do not do what you cannot tell;*" or, at least, if there be good reason for not telling, "Do not do what you could not tell without shame." If it is a question, whether you should take a commission or whether your employer should have it, ask him; and then there is no trouble. If it is a question of practice, doubtful as to ethics, see how it would look in print.

Lately there has been a reign of hysteria in ethics, and it has gone too far, no doubt. Nevertheless, without deciding whether this, that or the other thing was wrong, and while still believing in my own mind that many honorable men are being persecuted for having done things which only a short time ago all the world was doing, without such criticism, I may draw out of all this the one pervading moral. How many of these things, if they had been done openly, would not plague the doers now! Here is a railroad company, formed, after the American fashion, to drive its tracks through the wilderness, building cities as it went. And (horrible to relate!) it gave rebates to men who would help to build such cities. If the company had come out and said so, no one could have found fault justly. But it gave rebates and said nothing about it. In like manner, for many another thing, sometimes necessary, sometimes even wise, some hitherto reputable person or company has been made disreputable; and the real trouble is that the people who did it kept it secret.

There is another point about this matter of secrecy. When you keep a thing secret you are not very likely to make a complete record of it. Now, there is

nothing more foolish than not to keep your record perfectly clear. It is most humiliating to a man to be, after 30 years or so, unable to explain some particular payment which he knows was honestly made, simply because the stubs in his check-book do not tell the particulars. Keep your memorandum-books up to date and in good order, so that any one can see them. If you are ordered to do a thing which you do not approve, get the whole thing in writing, and make your record clear. We have been told upon the highest authority that men love darkness better than light when their deeds are evil. If you are not of such, why not walk in the light?

Professional Ethics.

BY VICTOR G. HILLS, DENVER, COLO.

(Pittsburg Meeting, March, 1910.)

THIS paper comprises suggestions on certain points rather than a complete survey of the whole subject. In other words, it is a contribution to the general discussion invited by John Hays Hammond in his paper, presented as a Presidential address at the Chattanooga meeting of the Institute, in October, 1908.[1]

Mr. Hammond says, in conclusion:

> " . . . I am not offering a code, or even an official and representative declaration of principles. My remarks are intended, and will, I hope, be accepted, simply as a contribution to that discussion of their theme in which every engineer of experience may participate."

Such a discussion has been conducted in technical journals, as well as in contributions to the *Transactions* of the Institute; and, in the remarks here offered, I shall have occasion to refer the reader to what I have thus published elsewhere, for more extended statements of my views.

What Mr. Hammond says about loyalty to employers, the double capacity of the engineer as an employer, the expert witness, commissions on machinery, the statement that most swindling is done by laymen and not by engineers, duty to the investing public, and the engagement of local engineers as colleagues, it is a pleasure to indorse. Under the head of the expert witness, Mr. Hammond refers to the address of Dr. R. W. Raymond on Professional Ethics.[2] This paper con-

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[1] *Trans.*, xxxix., 620 (1909).

[2] *Proceedings of the American Institute of Electrical Engineers*, vol. xxv., No. 11, p. 7 (Nov., 1906).

tains so much else that is relevant to this discussion that it is a temptation to make extensive quotations from it; but I will refrain from doing so, hoping that the paper may be reprinted by the Institute.

Mr. Hammond's "expert's Golden Rule"—"Tell unto others, now, whatsoever you would not have them tell on you (with unjust misunderstanding and scandalous comment) hereafter," is magnificent. Dr. Raymond puts the same idea in the epigrammatical form: "Do not do what you cannot tell." Both of these might be hung, as illuminated mottos, on the wall of an engineer's office.

Charles Catlett,[3] in his paper, Professional Examination of Undeveloped Mineral Properties, says much that falls appropriately under this head. His paper, both in what it declares and in what may be read between the lines, has a sound and elevated tone of combined professional honor and keen insight, which the active, thorough-bred engineer will not fail to note.

"Blue-Books," or Official Standards, for Engineers.

It has often been urged that engineers, like some other professional men, should have an official recognition of their ability and standing. Advocates of this idea would have a State license, or some plan by which there would be a legal recognition of ability, as there is with the legal and medical professions, the object being to protect the public. Exclusive societies sometimes go further and attempt to make their "blue-book" a guarantee of standing and character as well as of ability, and thus to protect both the public and their own members. This is, perhaps, beautiful in theory, but, for the most part, a failure in practice. The unscrupulous and indolent will develop the greatest energy of their lives in getting in, and then depend on official prestige to carry them. Let no one imagine that they will not get in. Attempts to keep them out are futile. Examinations avail but little, even in the line of ability; diplomas do not answer the purpose; "responsible charge of work," or other similar specifications, often admit a mediocre young man, whose rich uncle has secured him a salaried position ahead of a far more worthy independent pro-

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[3] *Trans.*, xxxix., 774 (1909).

fessional; recommendations and vouchers by those already inside, which should be the most effectual of all safeguards, are not availing; influence, social, political, or financial, secures indorsement with remarkably little trouble. Friends and relatives would rather get a weakling into some berth than to furnish him with employment themselves. An exclusive society may succeed in keeping its ranks up to a somewhat ideal standard while the organization is new; but, with increasing membership, divergent ideas and interests are bound to develop friendly cliques, which will either admit to membership those who are unworthy or exclude those who are worthy. Balloting by mail on unknown candidates is unsatisfactory. The ballot is thoroughly effective only with a local society, and even there a weakness is developed as the association increases in membership. This is demonstrated in the fact that the long-lived successful fraternities have numerous organizations of the same order in the same city. Among the older professional organizations, occasionally a bar-association, for instance, rejects an applicant; but, through the activity of influential friends, the candidate gets there next time. When a person is once in, it is seldom that sufficient evidence and energy can be developed to put him out. The medical profession is, probably, the most thoroughly and effectively organized of all. Its members are the most courteous to each other, and the organization is usually able to influence legislation in matters concerning its members; yet its best members will frankly admit that there are unworthy individuals among them.

Professional "blue-books" and "Who's who?" lists have hitherto failed in doing the very thing which they propose. A few of the utterly incompetent may be excluded by legal license-systems or by exclusive societies; but the artificial prestige given to those just able to work themselves in, opens another danger, by throwing the public off its guard; and this danger may be even greater than that of a free field, with the employer on the lookout.

Professional Timidity.

I once heard an engineer avow that he always took from 20 to 25 per cent. from his estimates of value, so as to be on the safe side. In his selfish caution he entirely overlooked the rights of his employer, who might be thereby robbed of a good bargain,

and of the examination-fee also. This reprehensible practice, or something like it, is so common that I have often felt constrained to say to a client that the results which I had submitted were exactly what I had calculated, and were as likely to be in error one way as the other. In some cases I have added a recommendation that he allow a certain percentage for safety in making his investment. In other cases I have said, "Your experience in investments of this character will enable you to apply your own safety-allowance without any recommendation from me." It is not an engineer's privilege to assume a safety-factor for a client without the client's knowledge.

Another form of cowardice appears in withholding the details which would enable another engineer, or perhaps an accountant, to verify the results given. It is not a sufficient excuse that the employer will not read the details if given to him. He may not wish to read them at the time; but he may find occasion to refer to them in subsequent operations. His money has paid for them; and the record he receives should include them. Sometimes the object of withholding data seems to be to make the author indispensable for future operations. It is doubtful whether this is often successful. Quite as often it has the opposite result—it acts as a professional boomerang and brands him as incompetent or unfaithful.

Court-Work.

More has been said and written on this subject than on any other branch of engineering work. The fierce criticism which has been brought upon the profession by the acts of some of its members in bitterly-fought law-suits has made this class of work better known to the general public. Some of this criticism has been just and some most unjust. Since there are always opposing attorneys with ability to make effective use of unjust and insinuating language, the weakest features of expert court-work have been thoroughly paraded.

Indorsing all that has been said by Mr. Hammond and Dr. Raymond, I have little to add. The idea that experts should be appointed by the Court, I am inclined to think is more fanciful than practical. Since the Court would have in advance no knowledge of conditions which would guide it in the selection

of the proper expert for the particular case, a certain bent of mind or the prejudice of an individual might decide a case unjustly.

Court-work is unpleasant to most professional men. I recall only two engineers whom I have heard declare that they like it; one because his bellicose nature enjoys the conflict with the cross-examining attorney, and the other on account of the mental discipline experienced by him under such examination.

Vendors' Reports.

I agree with Mr. Hammond that a report for a seller should not be condemned as unprofessional. Indeed, he goes perhaps too far. If proper precautions are used, I hardly see why even the young engineer need avoid this work. When such a report is made in the proper form, its character as a seller's description of his property is apparent, and it frankly challenges verification by any other engineer who may be selected by the purchaser. The author protects the public in protecting himself. A mistake will reflect on his judgment, and a too-enthusiastic description will react injuriously on his business, but not on his honor. The fact that a report is made for the owner should show in the text and over the signature. If the engineer is to be compensated for his work by a commission, he makes himself a partner in the enterprise, and this must also be shown in the report. Then his statements will be weighed accordingly. The small investor will buy with his eyes open, and the heavy investor will, in any event, employ his own engineer.

Professional Confidence.

One of the vital principles of professional ethics is the inviolability of confidential information. This is generally recognized by engineers and by all professions, but there cannot be too emphatic a declaration of it. What Mr. Catlett says on this point is good and thorough, and I fully indorse it. But complete confidence on both sides is essential to a satisfactory result. The client who, keeping his plans to himself, engages an engineer simply to give him information, is really not a client, but only an employer, and does not realize the enormous handicap which he is placing upon his employee. The time and labor involved in making a report may be doubled or quadrupled for lack of knowledge of the exact purpose for which it

is to be made. The engineer, for fear of overlooking some point which might prove important, does unnecessary field-work, and so on through his whole engagement, wasting ammunition by firing in the dark. Even then he is likely to miss something which would have been desirable. With full mutual confidence, the work is done more quickly, economically, and satisfactorily. By explaining this, I have usually induced my client to state his purpose more fully. In my younger days, I used to wish that he would stay away from the work until it was done; now I rather encourage him to accompany me. However uncommunicative he may intend to be, he will be likely to reveal in conversation more than he would have put in writing; and thus I am enabled to serve him more effectively.

Stock Speculation.

Under this head I include the buying and selling of property interests, whether in the form of stock or not.

J. R. Finlay says:[4] "As to an engineer investing in stocks or speculating in them, it seems to me that his activities in that direction should be no more limited than those of any other man of common sense." I have already expressed elsewhere[5] my dissent from this proposition, so far as it applies to the buying and selling by an engineer of the stock of a company with which he is associated professionally. As I then said: "The limit is that professional engagement and pecuniary personal speculation shall not be tolerated together in connection with the same enterprise;" and I added that, apart from such professional connection, the engineer has an unquestionable right to speculate in stocks, though I questioned the policy of his doing so. For a more extended statement of my views on the subject, I would refer to the article cited above.

Contingent Fees.

Contingent fees furnish the chief debatable ground in the discussion of professional ethics. If an engineer should make a report for a seller, for which he is to receive a greater amount in case the report accomplishes its object, or for which the compensation is a commission on the sale, the performance is one

[4] *Mining and Scientific Press*, vol. xcviii., No. 2, p. 80 (Jan. 9, 1909).
[5] *Idem*, vol. xcviii., No. 9, p. 312 (Feb. 27, 1909).

thing if the relations are kept secret, and quite another thing if the relations receive full and unreserved publicity, by which the transaction is at once transferred to the category of "vendor's reports," and would be practically so considered by any buyer. The author of such a report, having announced himself a partner of the vendor, has satisfied the requirements of professional honor, and henceforth, in that particular transaction, is amenable only to the rules of common sense, honesty, and policy. A vendor's report, made for a fixed compensation, with suppression of the fact that it is made to promote a sale, would be reprehensible, but it would not come under the head of "contingent fees."

Mr. Hammond, in justifying contingent fees, apparently confined his opinion to vendor's reports, publicly acknowledged as such, or those which are virtual partnership arrangements. So far his declarations are sound and safe; but his proposition that a legitimate contingent fee should depend upon "the subsequent success of the enterprise" seems to me impracticable. How long shall we wait to satisfy this condition? and, in case of failure, how settle the question of the responsibility?

In this connection, however, I would add that a contract for the receipt of a percentage of the proceeds of a proposed metallurgical improvement is, in my judgment, an excellent and unobjectionable professional partnership.

Removing from the category of "contingent fees" the cases which are virtually partnerships, the problem is simplified, and there remain only the cases in which something is received as remuneration which may depend for its value, wholly or partly, upon the color of the engineer's report; in which personal gain might be, or even might be conceived to be, a temptation to say a thing or to leave a thing unsaid. In such instances I contend that a contingent fee is unprofessional and culpable; that an engineer has no more right to accept one than a judge has to sit on a case in which he is interested. A judge, however high and unimpeachable his position, will decline to sit where a relative or near friend is interested. The engineer's position is quite as delicate.

If a "contingent fee" were put in such a homely form that it could not possibly escape being branded as a bribe, even our "commercial" engineers would regard it with scorn. And if

we could class all cases as either bribes or partnerships, the question would be settled; however, the painful fact confronts us that a goodly number are found defending contingent fees by sophistry and labored illustrations.

Forms and Contents of Reports.

The days are long past when elaborate treatises on history, geology, and topography were used for the body of an engineer's report; but the swing of the pendulum seems to have brought some to the extreme of presenting seductively simple and summary financial statements only. Mr. Finlay has given[6] typical report-forms illustrating the extremes, and advocating the "Brown" form, thus:

> Mr. Smith, engineer: ". . . I have examined Mr. Jones' farm of 160 acres in Washington county, Pennsylvania. I find such and such openings and such and such drill-holes. I estimate that the coal seam on the farm will average 5 ft. 9¾ in. thick, and that the average analysis of the coal is 60% fixed carbon and 10% ash, etc."
>
> Mr. Brown (another engineer): ". . . Jones' farm of 160 acres can be depended on, in my judgment, to produce a million tons of coal, which can be mined for $1 per ton and can be sold for $1.25 per ton. It can be worked out in 10 years, at a total profit of $250,000. I figure its present value at $125,000."

Neither of these reports is satisfactory. The "Smith" style is antiquated; the employer wants the figures in money-value, or else he would not employ an engineer at all; but the "Brown" report is equally bad, professionally, and more dangerous; because the employer could take the Smith report and, perhaps with some assistance, arrive at a solution of the commercial problem. A proper report should contain the items given by both Smith and Brown, and more. There should be a map showing the location of every opening and drill-hole and the thickness of the coal at each. In short, the report should contain all the information employed in reaching the author's conclusion, and paid for with the employer's money. Statements of quantities and values are usually not enough; they should be reinforced by opinions, based on experience and judgment, including such matters as probable changes in transportation and market-supply, which are likely to make the whole difference between profit and loss.

A report consisting almost wholly of opinions is another weak

[6] *Mining and Scientific Press*, vol. xcviii., No. 2, p. 80 (Jan. 9, 1909).

form. The conscientious engineer will not assume omnipotent judgment and exclude the employer from his confidence, by withholding the data and the reasoning which underlie his conclusions. Moreover, it is his duty to give such details as will enable a subsequent expert investigator to check and weigh the opinions he has expressed.

While pedantic exhibitions of scientific knowledge are to be condemned, the subjects of geology, mineralogy, and the like should not always be ignored. Instead of omitting all mention of geology, I think it usually best to give something, probably a plain condensed statement, in a single paragraph. The mineralogy may be dispensed with, or it may be disposed of in a few words, or it may be the main feature of the report—and so on with other items, according to circumstances.

To avoid making a report too cumbrous and dull, I like the plan of putting a table of contents, together with an abstract and summary, at the beginning, instead of a recapitulation at the end. Such details as assay-lists, detailed plats, tabulated cost-statements of various kinds, etc., can be put in supplements or appendices for convenient reference. The preliminary abstract, probably condensed to a single page, with the explanation that it contains the essence of the entire report, and that the remainder is submitted for the sake of completeness and possible future reference, will make the report satisfactory to any reader and under varying conditions.

It should not be understood that I advocate elaborate detailed reports in all cases. One of the most important and satisfactory reports which I ever made covered about four sheets of note-paper, and was written with a fountain-pen while riding on a railroad train; another had to be made, practically complete, by telegraph. The circumstances vary widely in a mining engineer's experience.

It is hardly necessary to remark that an engineer should be scrupulously careful to keep an exact copy of every report, including the plats as well as the text, as a protection. To prevent the mutilation of a report, I often write, just over the signature, something like this: "Five pages of text, three plats, and two inventory-sheets." It is always possible that a report will turn up at a time and place for which it was not intended. This cannot be avoided. The recipient has the right to publish or

sell a report, or to make any use of it which he sees fit, so long as he does not make it misrepresent itself by mutilation or unfair quotations. A little care in embodying, to some extent, circumstances, conditions, and purposes in the text of the report, will do much to forestall future misrepresentation or misuse of it.

The Demand for Sufficient Time.

There is no section of this subject more worthy of consideration than that of demanding sufficient time to do justice to professional work. The young engineer in particular can study this subject with profit, but the question never vanishes throughout the longest professional career. The engineer is continually approached to make some report, or to act as an expert witness, without sufficient time for preparation. Sometimes, from shrewd motives of economy, the employer comes with the statement that the work must be done by a certain date; sometimes the cunning promoter endeavors to cut the time short in order that the weak points may escape observation; but perhaps more often than otherwise, the employer, through pure ignorance of the time necessary, delays speaking to the engineer until the expiration of an option or the date of a court trial is too near at hand. There are cases where professional aid is sorely needed in like emergencies, and the engineer may respond to the call, and is constrained to do so, particularly if it is for an old client; but such work should be undertaken only by those of experience and fairly well-established reputation, and avoided as an extremely dangerous class of work by the inexperienced. The temptation is great; it is comparatively easy to stay the conscience with the thought that an engineer cannot report on more than he is able to see, or be held responsible beyond the points covered. Nevertheless, right here lie great dangers and many pitfalls. If he cannot do justice to the case he cannot do justice to himself. He must decline employment, to stand by his colors. I can think of no branch of the subject in which professional ethics comes closer to every-day practical application.

Complications and Embarrassments.

An engineer is called upon to make an examination and report where he already has much knowledge gained from work

on neighboring properties. Perhaps it was this knowledge which led to his selection. He cannot betray the confidence of his former employers, and he is practically unable to act and judge entirely independent of his previous knowledge. Sometimes permission to use any data will be freely granted by the prior employers; sometimes the work can be done in an entirely independent manner without embarrassment; sometimes the offered employment is a covert attempt to bribe, an attempt to secure some knowledge of a neighbor's property which could not be obtained in any other manner; and, most embarrassing of all, this object may not at first be apparent, or the engagement would have been declined at the outset. Between these extremes there is a field for every possible shade of embarrassment. The situation changes by insensible gradations from cases which would be promptly declined by any self-respecting man to those which could be performed with the most frank and agreeable understanding of all concerned. Here no fixed rule can be established, and common sense and discernment must be the guide to keep on the safe side.

Using the notes and data taken in a case where the question of secrecy is waived, or not involved, and charging for the same in future cases, is an often-discussed question. I can see no difference between notes on paper and those which are impressed on the mind. The older professional man is expected to earn more with accumulated knowledge. The question may be a matter of bargain and understanding with the client.

The right to sell information to another, where the first employer has hopelessly failed to pay his bill, raises a fine point; and this is further complicated if a small advance payment on account has been made. The law and the equity might conflict. A case might depend so much on the particular conditions that no general rule could be made applicable.

Taking advantage of knowledge gained in the employ of one mine to secure a personal interest or a lease in an adjoining property, raises an ethical point. By first imparting the full information to the employer, and giving him the refusal of the same opportunity, the way is cleared. Holding a lease on a property which pays the engineer a fee for consultation may possibly be justified in rare instances, but, even with full knowledge, it is, to say the least, treading on dangerous ground.

An engineer may be employed at two neighboring mines; after a while there comes an unexpected clash of interests, when he is expected to advise both parties. Then he can do neither more nor less than stand aside and tell the contending parties to employ other experts. And the same idea may be carried out when the engineer's personal interest conflicts with his employer's. Both attorneys and engineers are sometimes caught in such a predicament: it may involve embarrassment, but can always be so handled as to avoid dishonor.

The professional members of the United States Geological Survey are not only forbidden to testify in any mining-litigation during the term of their employment, but are bound not to do so for three years after leaving the service. The reason for this is sound and obvious. In general, the engineer should not serve another client too soon after quitting the employ of a rival. How soon, must depend on circumstances: often he can never honorably do so.

THE LIFE-LONG PROFESSION.

Granting an engineer's right to speculate in stocks, to become a partner in purchases, or to act as a promoter when not in connection with a professional engagement, there still remains the question of good policy, involving all the really fine points of professional ethics.

If we are to follow the profession for life we must forego some of the rights of the commercial engineer. It is a mistaken idea that the employer of professional service looks for one who is himself exhibiting great personal financial prosperity. My observation is that too keen an exhibition of the money-making quality rather begets distrust. The miner or the investing public is more sensitive to professional constancy than some seem to perceive.

Even when considered with reference to pecuniary income, the question should be treated from the stand-point of a business which is to yield a revenue throughout an active life, and not as only a stepping-stone in finance. We do not have to reflect long to recall some engineer whose financial activity readily betrayed the fact that he had taken hold of the profession only as a vaulting-pole, to be quickly dropped when it had served its purpose. The commercial engineer is like a lessee,

who sinks a little shaft, without timbering if possible, leaves rubbish in every drift, gouges out by underhand stoping the first bunch of ore found, and finally leaves the place to cave in. The thoroughbred professional man is like an owner who puts down a well-timbered shaft, of sufficient size and equipment, and does his work in "miner-like manner," that the place may be worked as long as it will yield. Our commercial engineers at least lay themselves open to the suspicion of treating the profession after the manner of the lessee. In discussing the ethics of the situation, I think we should take the stand of the owner and ignore the lessee.

A Portable Assay-Outfit for Field-Work.

BY S. K. BRADFORD, NATIONAL. NEV.

(Pittsburg Meeting, March, 1910.)

For years past I have traveled in quest of promising mining-properties, over almost impassable mountain-trails to remote places in the mining-regions, usually many miles from an assay-office.

If, upon examination, the formation and the ore-deposit appeared favorable, a quick determination of the value of my samples was necessary, in order that I might take intelligent action with reference to the acquisition of the property. A journey with my samples to civilized regions often meant weeks of delay, and perhaps loss of opportunity. How many, under similar circumstances, have returned with their assay-results from such a journey, only to find that some audacious speculator had "optioned" the property! True, a first payment could have been made and the chances taken on the results; but experience has taught that rich-looking ore is not of necessity valuable. Personally, I would not hazard a ten-dollar bill on any property without first determining by assay the value of the ore.

To carry the usual complete assay-paraphernalia over such mountain-trails was impracticable by reason of the loss of time involved, the great expense of transportation (charged accord-

ing to weight), and the fact that a few such trips would almost certainly wreck the outfit.

For these reasons I began a process of elimination and substitution which finally enabled me to carry, in an ordinary 26-in. valise, sufficient apparatus and supplies for, say, 100 assays of adequate commercial accuracy. My final outfit was as follows:

Apparatus.

Small hammer; steel mortar (2.5 in. wide, 3 in. high, with stand sawed off flat, so that it can be used as an anvil and for slag-molds); small steel pestle; blow-pipe and blow-pipe cupel-holder; small spatula; large spatula; button-pliers; light 1-in. cupel-mold; two 3.5- by 8-in. muffles; 18 "F" crucibles and 6 crucible-covers; short crucible-tongs; 25-cc. burette; six 0.5-oz. Denver parting-bottles; six 0.5-oz. Berlin annealing-cups; two or three beakers; hand-balance, such as is used for weighing gold-dust (folding in tin box), and set of gram-weights; a 6-in., 60-mesh screen; 6 small glass stirring-rods; teaspoon and tablespoon of usual camp-style; and a micrometer for measuring the diameters of buttons. (This may be a short, fixed microscope-stand, in which has been placed a scale of 0.1 millimeters—two or three extra object-glasses being carried for safety. If one is willing to be burdened with the extra weight and bulk, a small portable button-scale and weights may be substituted. The micrometer, however, is sufficiently accurate, and possesses the advantage that if one's mule should roll down the mountain it will not be injured; whereas a scale may be hopelessly ruined for further immediate field-use.)

Chemical Reagents, and Supplies.

(To be carried in bottles inclosed in metal cases): 0.5 lb. of potassium cyanide; 1 lb. of C. P. nitric acid; 1 lb. of C. P. hydrochloric acid, and 2 lb. of aqua ammonia.

A few ounces of sheet-copper; 1 oz. of C. P. sheet-silver; 0.5 lb. of sheet-lead; 2 lb. of ten-penny wire nails; 5 lb. of bone-ash; 5 lb. of sodium bicarbonate; 5 lb. of litharge; 3 lb. of red lead; a few pieces of prepared charcoal for blow-pipe

use; half a dozen miners' candles and half a dozen old newspapers.

Practicable Substitutes in the Field.

Sodium bicarbonate can be obtained at most country stores. If the supply of litharge be exhausted, red lead (used in the proportion of about 3 assay-tons) makes a fair substitute. For a long trip I increase my supply of bone-ash and litharge. Under extreme conditions I have been compelled to get bone-ash by calcining and pulverizing old bones, and to substitute powdered beer-bottles for borax, filings from coins (of known fineness) for silver, and flattened bullets for sheet-lead. If, as is not unlikely, the supply of beakers should have been destroyed *en route*, beer-bottles with the tops broken off constitute a tolerable substitute—and, I need hardly say, can be obtained in abundance at all mining-camps!

The outfit above described can be packed in an ordinary valise, say, 26 in. long. To illustrate its use in the field, I present the statement which follows.

Gold- and Silver-Assay.

The ore-sample is broken to very small size, quartered, and passed through the screen. The final assay-sample (0.5 A-T., unless the ore be very poor, but up to 2 A-T., if necessary) is weighed in the gold hand-balance. To a sample of 0.5 A-T., add one even teaspoonful of litharge. If the ore is very heavy with sulphides, 5 even teaspoonfuls of red lead and 10 nails may be substituted. To this should be further added a "heaping" tablespoonful of sodium bicarbonate, and another of borax, for which, if the ore is very basic, ordinary powdered glass may be substituted, or also added. The final addition of 0.5 g. of wheat-flour or powdered charcoal will give, after melting, a lead-button of from 7 to 8 g. weight, unless the ore contains a large proportion of heavy sulphides. The sample should be thoroughly mixed, wrapped in newspaper, and properly numbered for future identification. If the ore be believed not to carry sufficient silver for successful subsequent parting, a small measured portion of C. P. silver may be added.

For crucible-melting, a blacksmith's forge is available in every mining-camp. In using such an apparatus for this pur-

pose, a small inclosure of brick or loose stone should be placed around the tuyere of the forge, built up nearly even with the tops of the crucibles, and a layer of coal or coke should be put directly over the tuyere on which the crucibles stand. The spaces between the crucibles and the walls should be filled with fuel. The crucible and covers (or, in case of muffle roasting, the muffle) should be thoroughly dried and warmed beforehand, to prevent cracking. The charge having been put into the crucible, with a layer of borax on top, the crucible cover put on, and the whole covered with charcoal, it is only necessary to take care not to make too hot a fire (thereby melting off the bottom of the crucible) and to turn the crucible from time to time, in order to secure an evenly-distributed heating. During the roasting-process, the crucible-cover should be removed.

If the fluxing was correct, the melt will pour well, and the crucible can be returned to its place in the fire, and the next charge, paper and all, put into the hot crucible. There need be no fear that the crucible will "salt" the subsequent assays.

With care one can make from 6 to 18 assays in each crucible. Fuel must always be kept between the bottom of the crucible and the tuyere-iron. Unless the samples are very base, from 15 to 30 min. will be sufficient, after the first melting, to melt each lot of four.

When all the samples are melted and ready for cupellation remove the stone or brick around the tuyere, and place on each side of it a brick (or a flat stone of about the same size) the two being about 6 in. apart. On these place the muffle forming a bridge over the tuyere, with the mouth of the muffle facing the front of the forge. In using stones for this purpose care must be taken to select those that will not spall under the heat and break the muffle. Set the muffle level and build a wall around it, excepting a front space about 3 in. away from the muffle, and to about the height of its top. Cover with charcoal, and keep plenty of fuel always under the muffle on the tuyere. Put in the empty cupels and close the mouth of the muffle with a piece of charcoal or coke whittled to fit it. After the muffle has become hot put in the buttons.

In the absence of charcoal or other suitable fuel, I have

made charcoal in an open fire or fire-place by a liberal use of water to cool it or by covering the charred fuel with dry dirt.

The above method of fire-assay requires less fuel than any charcoal-furnace I have ever used; and I have often made, in this crude way, 40 assays in a day. The buttons come out clean and bright; and the cupellation requires but a few moments longer than in a regular cupeling-furnace.

Sometimes the muffle breaks, and must be set up on pieces of iron running lengthwise, and plastered with clay to hold it together. When a muffle is entirely gone beyond such patching, and no other is to be had, a new one may be drilled out of a brick. But I have often made 20 runs with one muffle.

Having cupeled all the buttons, one should find some place out of the wind and, if one has the scale, part and weigh the gold in the usual manner. If using the micrometer, measure the diameter of the button and determine its weight by a previously-prepared scale based on specific gravities, multiplied by solid contents. This method of determining weights was published[1] by Luther Wagoner, of San Francisco, some ten years ago. After parting, the gold is dried, wrapped in a small piece of sheet-lead and cupeled with blow-pipe and miners' candle on a little bone-ash pressed into a cupel-holder. The button being perfectly round, the weight is determined by measuring the diameter, as described. The cupel-holder is made from a piece of volcanic tufa cut down to a cylinder in form, about 1.5 in. in diameter, and 2 in. long. The top is hollowed out like a bowl about 0.5 in. deep; and a small hole, 0.5 in. long and about 0.25 in. in diameter, is cut in the bottom. Into this hole fits a piece of wood, which the assayer can hold in his left hand, so as to turn the cupel-holder with his thumb and forefinger, and make a perfect button. The tufa having been heated in a fire to a red, nearly white heat, and then dropped into water, comes out as hard as a rock.

A few years ago, I forgot, on one trip, to take with me the pulp-scale and weights; nor did I discover the omission till I was ready to weigh the pulp, in a locality more than 100 miles away from the nearest town. I melted the tops from two "Carnation" cream-cans, and punched three equi-distant holes

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[1] *Trans.*, xxxi., 800 (1901).

through the rim of each top and put in strings of equal length, thus making pans for a scale. Then I put a common dressing-pin through the middle of a thin strip of pine, and another through each end, at equal distances from the first; balanced the stick; hung my pans to it; and having balanced these, was ready to weigh the pulp. I had a worn half-dollar silver piece, which, with a new ten-cent piece, would exactly balance a new half dollar, weighing 205 grains. Then, by adding a 5-cent silver coin, I had 230.625 grains, near enough for my purpose to 0.5 assay-ton.

If one doubts the existence of any commercial value in the ore, one may carefully weigh out 0.5 g., run it down on a piece of charcoal, and measure the button with the micrometer, even though it be so small that the natural eye cannot see it. In this way, the trouble of making a regular assay may often be avoided.

Copper- and Lead-Assays.

For volumetric copper-assays, it is necessary to standardize a solution at each camp; since it is impossible to transport the solution without standardizing it when next used. Therefore it is best to throw away the old solution when packing, and make a new one when required. Often I make rough fire-assays of copper, to determine whether it is worth while to investigate the copper-value more thoroughly. This can be done in connection with melts for gold and silver.

Lead can be determined in the same manner. An excess of carbon should always be used in the charge for lead. It is advisable to keep a duplicate sample of each assay, and have it checked on the return to civilized regions. After one has become familiar with the above plan of assaying, however, the testing by duplicate assays of such preliminary determinations of probable value will soon be discontinued.

The Gold-Fields of French Guiana, and the New Method of Dredging.

BY ALBERT F. J. BORDEAUX, THONON-LES-BAINS, SAVOIE, FRANCE.

(Canal Zone Meeting, November, 1910.)

I. GENERAL INTRODUCTION.

1. *Historical.*

ALLUVIAL gold was first discovered in Guiana in 1852, in the sands of the Arataye river, by Paulino, a Brazilian convict. During the following years, gold was found also in the rivers Orapu, Cirubé, etc.

In 1873 the famous placers of the Sinnamary system were opened: Saint-Elie, Dieu-Merci, Adieu-Vat, and Couriége. This group of mines has produced nearly $10,000,000, according to official records, not including the gold that escaped the taxes. The discoverer was a mulatto named Vitalo. Except in a few rich spots, the gold was rather evenly distributed along the creeks. In one instance, gold was found in the red dirt from the slope of a hill, which, after transport to the creek, gave an amount of 150 kilograms.

The placers of the Mana system, Enfin, Elysée, Pas-Trop-Tot, date from 1878, and their production has been about $5,000,000. In 1888, the Awa placers, along the boundary between the French and Dutch Guianas, on the Maroni and its tributaries, were discovered. They have yielded $12,000,000. The only industrial company, the Dutch Guiana French Mining Co., produced in 15 years about 2,500 kg. of gold, worth $1,500,000.

From 1893 dates the famous Carsewene discovery, in territory then in dispute between France and Brazil. A settlement was made recently in favor of Brazil, by arbitration of the Switzerland State Council. Carsewene is said to have produced about $18,000,000 in six years (1894–1900), but not even one-fourth of the production went through French Guiana and

paid taxes; the balance went to Brazil and the West Indies and can hardly be estimated. The gravel was very rich, but patchy. Some places, washed over repeatedly, could not be exhausted. An industrial company which built a mono-rail railroad, came too late, and got very little gold. After the set-

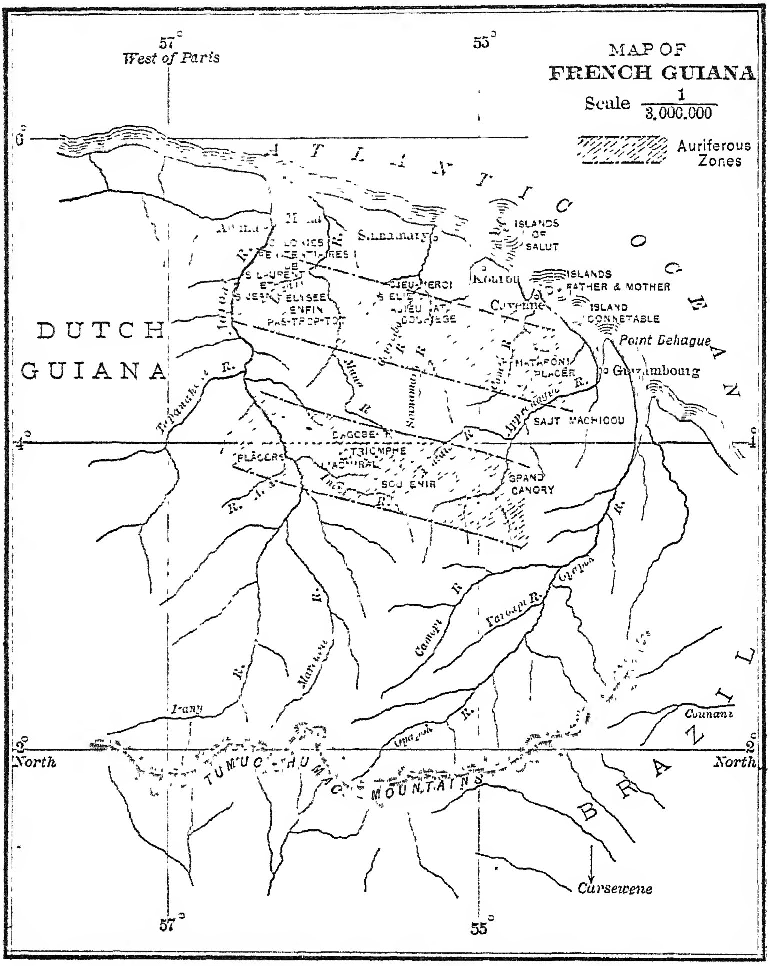


FIG. 1.—SKETCH-MAP OF FRENCH GUIANA, SHOWING AURIFEROUS ZONES AND MINING-DISTRICTS.

tlement, the miners abandoned the place, which is now to be tested by dredging.

In 1901, the Inini gold-discoveries attracted a rush of miners. The gold was very irregularly distributed, large nuggets alter-

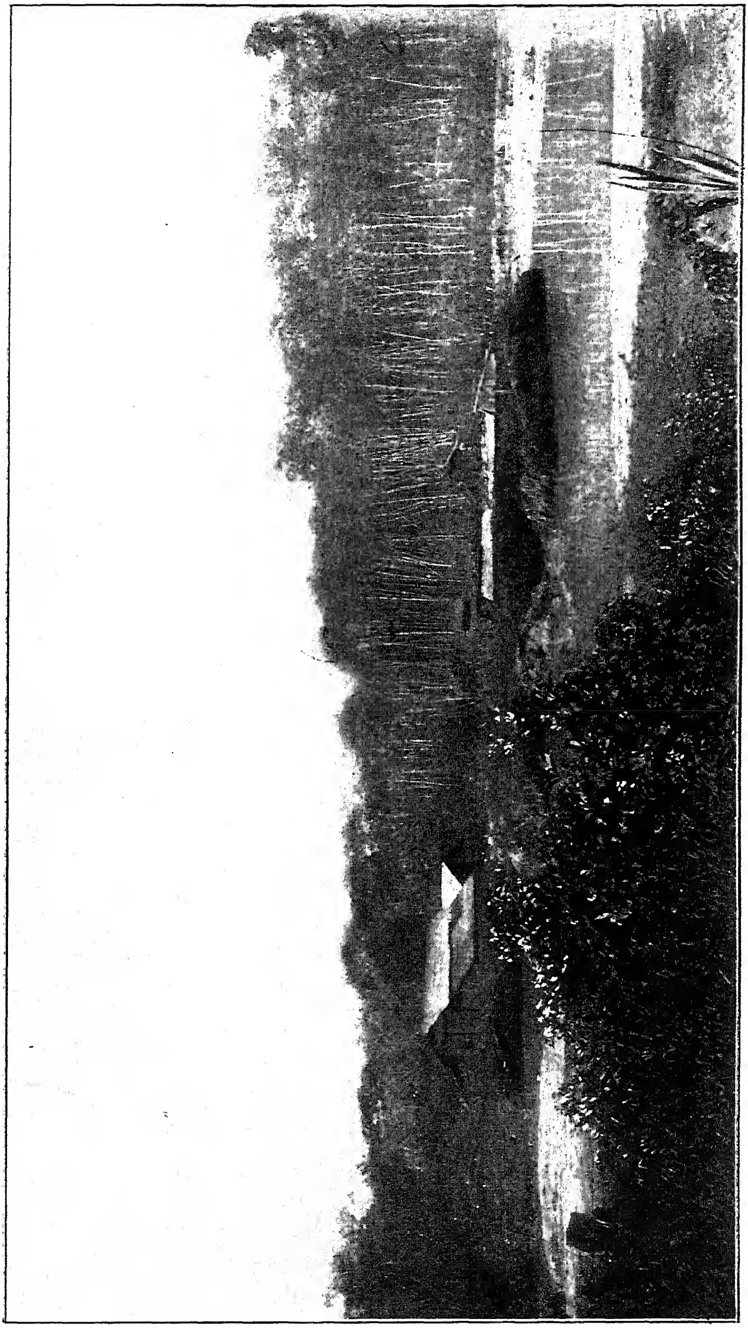


FIG. 2.—TYPICAL GOLD-PLACER AND DREDGE, FRENCH GUIANA.

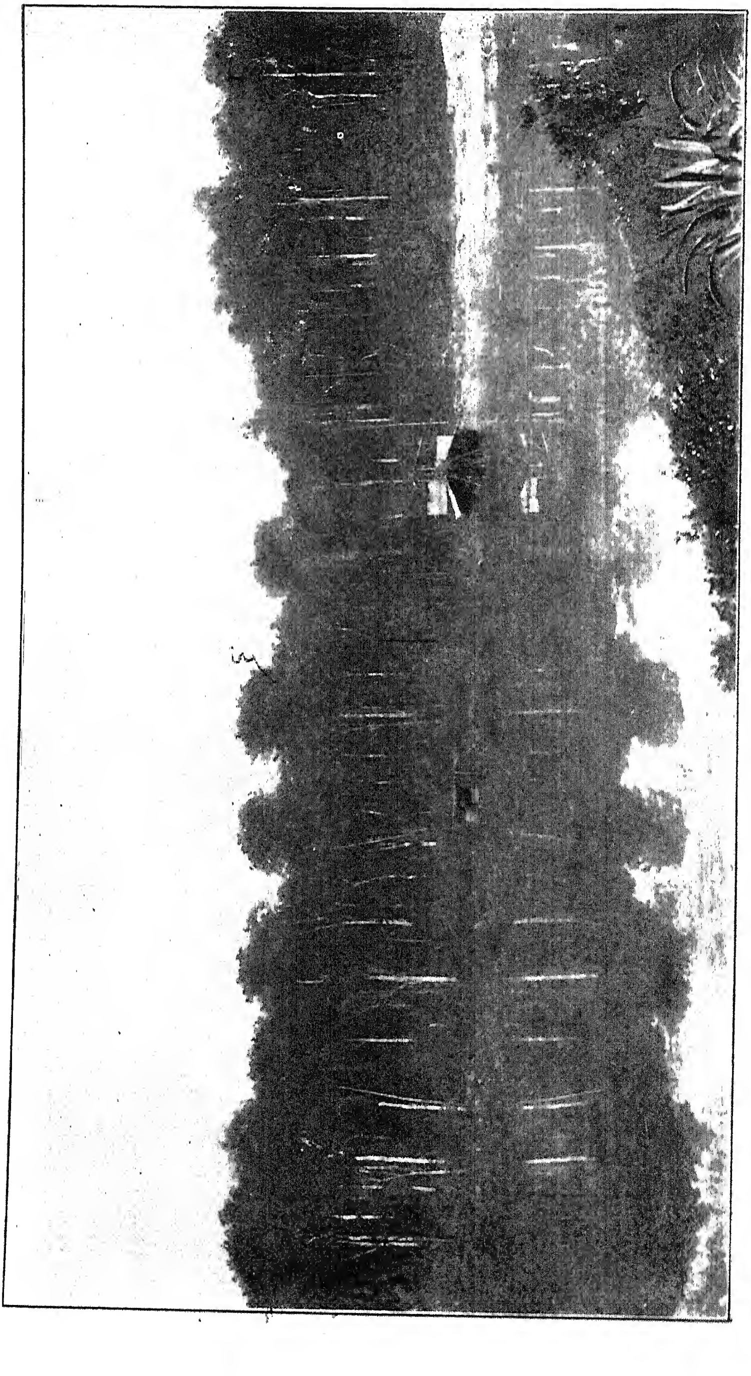


Fig. 3.—Elysée Placer, French Guiana; Gold-Dredge at Work.

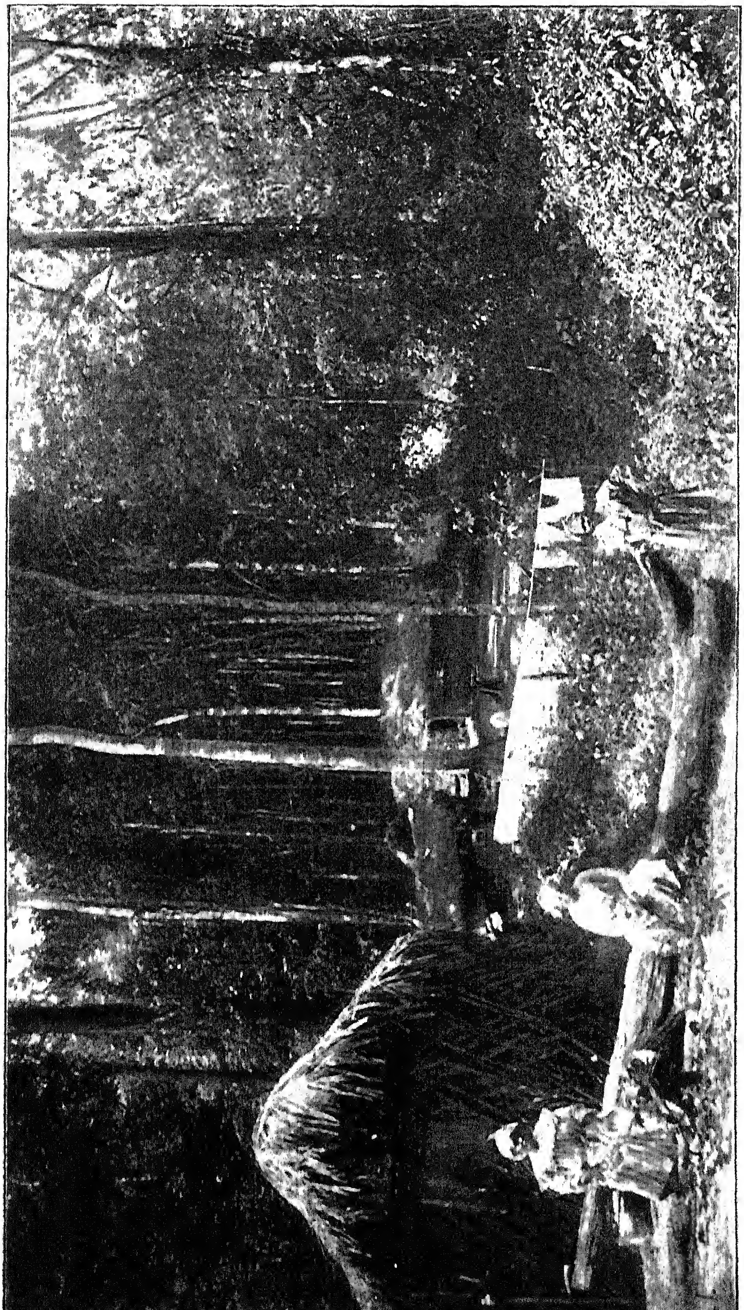


FIG. 4.—TYPICAL FOREST VIEW, LEZARD CREEK, FRENCH GUIANA.

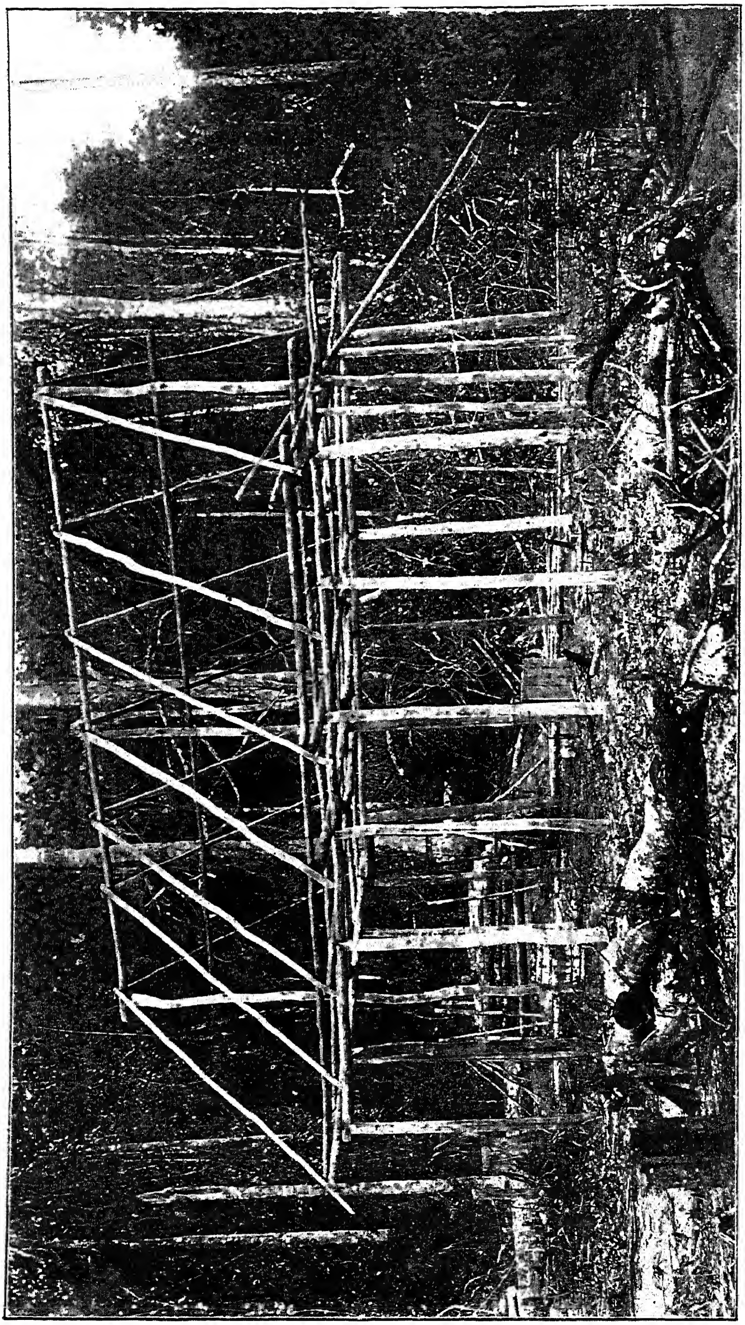


Fig. 5.—Native Framework, Construction of Building, French Guiana.

nating with barren zones. The production was perhaps \$4,000,000. Some of the Inini nuggets were famous, but they had to be cut up into shares for the miners.

The most recent important discovery, made in the upper Mana river in 1902–1903, yielded in three years more than \$3,000,000.

The Kokioko discovery in 1907 was a very small spot on a tributary of the Mana river, but this and another on the lower Maroni may lead to the development of a new kind of deposits—an old river-system.

Fig. 1 is a sketch-map of French Guiana, showing the auriferous zones and mining-districts. Figs. 2 to 5 are typical views in placer mining-districts.

2. *Production.*

Statistics are somewhat deficient; they had no existence before 1866, and the sole records are the custom-taxes. But as the gold may easily escape to Brazil, Dutch Guiana, or the British Islands, where the taxes are much lower (5 per cent. in Dutch Guiana against 8 per cent. in the French), or more difficult to collect (as in Brazil), the official figures are doubtless largely below the real output. They are as follows by decades:

	Kilograms.	
1866–1876,	7,368.060	
1876–1886,	18,446.379	
1886–1896,	20,795.722	
1896–1906,	31,081.485	
1906–1910,	16,000.000	estimated in 1909.
Total,	93,691.646	

The value of this gold is about \$56,000,000. But if we take into account the free-trade gold, we can nearly double the figures, making, say, \$100,000,000.

For instance, from Carsewene, a small proportion only, as I have said, went through Cayenne. The Carsewene was indeed an exceptional discovery. That and the St. Elie are the most remarkable in the Guianas. According to old reports, it was the famous Eldorado already known through Christopher Columbus. It is well known that neither Columbus nor Cortez ever struck gold in the Indies and Mexico. They found mines

of silver only. The gold which they got from the natives came from remote countries, between the Orinoco and the Amazon, for which the Spaniards looked in vain.

3. *Geology of French Guiana.*

The formation of French Guiana cannot yet be minutely described, as the country itself is not well known. The ground consists chiefly, for a depth of from 40 to 100 ft. and more, of a decomposed rock of a yellowish or reddish color,—laterite and limonite (*roche à ravets*, ferruginous and cavernous rock). The elevations are very small, and hence the outcrops of veins are scarce. The formations cannot be detected except by sinking shafts; and, since there are practically very few such works in Guiana, our knowledge of its geology is limited.

Nevertheless, some outcrops of rocks may be determined, first along the sea-shore, and secondly along the rivers, where they show a water-fall, or a *saut* (jump), as the creoles call it. During the transport of canoes and their cargoes along the *sauts*, an examination of rocks without loss of time is possible.

The rocks so far known to exist in French Guiana are the following, for the most part crystalline or pre-Cambrian, always without fossils:

Granitic gneiss, amphibolites, mica-, talc- and clay-schists.

Common granite, diorite and diabase, and melaphyre.

Quartzite, fine-grained sandstone, and quartz-veins.

Limonite, with a distinct cavernous or cellular variety (*roche à ravets*).

The rocks in the rapids (*sauts*) of rivers are mainly quartz-veins, or dikes of granite or melaphyre. They always follow the same direction, which suggests regular stratification, or regular faulting.

In the above rocks, the following minerals have been detected:

Gold, in quartz-veins and quartz boulders disseminated in the laterite of the surface (at Saint-Elie, Adieu-Vat, Elysée, etc.).

Silver, as silver sulphide at the Silver mountain (near the mouth of the Oyapok), prospected by the Dutch between 1652 and 1658.

Copper and lead (slightly prospected).

Tin, below the Tumuc-Humac mountains, associated with

iron, manganese, and even quicksilver: but no serious work has been done.

Coal is said to exist near the Maroni.

Phosphates of alumina are strenuously worked by an American company in the Connétable Island.

Diamonds have not yet been actually discovered; but they exist in similar formations in English Guiana.

II. The Gold-Bearing Gravel.

The gold-formation comprises two general East and West zones, one from 40 to 50 km. wide, and from 50 to 100 km. from the sea-shore, and the other along the southern part of the upper Mana and Maroni rivers.

1. *General Description of the Placers.*

The pay-streak is the lower stratum in the gravel of numerous creeks, and their junctions with main streams. These creeks run between undulating hills, rather steep at the base. The width of gravel in the upper creeks is seldom more than from 5 to 8, and never more than from 10 to 20 yd. Windings are frequent, the grade of the streams being very gentle. In the same creek, gold may be found along 4 or 5 km., and even more.

The quantity of water varies greatly, according to seasons. The small creeks carry from a few to more than a hundred liters per second; the main creeks, several thousand liters per second. The natives call them summer or winter creeks, according to the possibility of washing the gravel during the summer or the winter. As a rule, the main creeks, having much water, may be washed during the summer, and the small creeks in winter, being often dry in summer. Hydraulicking is out of the question, by reason of the flatness of the ground.

To a large extent, the gravel in all creeks is the same; the pay-streak, mainly composed of quartz sand and felspathic clay, is directly below a barren zone, consisting of clay-dirt, intermixed with tree-stems and roots.

The profitable working of a placer depends chiefly upon the degree of its accessibility. But, as a rule, the upper creeks running along the rich zone are better than the lower ones.

The gold nuggets are sharply ended, proving the immediate vicinity of the gold-bearing rocks. Large nuggets are seldom found in the upper Mana creeks, but have occurred often in the upper Maroni and Inini tributaries.

A great drawback in the placer-workings is, that the upper seam of dirt is much entangled with fallen trees and their roots. In thickness, this layer ranges from zero to 7 and even 12 ft. in the big creeks. The gold may be found under the tree-stems, or under boulders, which must be blasted. If the necessary operations cost too much, the place is abandoned. Dredging under such circumstances is, of course, not to be thought of.

The bed-rock is a decomposed granitic or gneissic rock, soft and gently undulating. Some remnants of this rock, less decomposed or even intact, may be found (but seldom) cropping among the hills.

The hills dividing the creeks are also of decomposed rock or reddish laterite, sometimes intermixed with quartz boulders, which are either scattered at random, or follow a certain direction on the surface, but disappear in depth. In a few places, former quartz-veins have fallen sideways as the country-rock was disintegrated, thus losing all connection with the rock in place; and the same may be the case with large boulders of granitic rocks resting on the laterite. At St. Elie, an old quartz-vein was folded like an accordeon. Elsewhere (at Carsewene, etc.) there is to be observed a general direction of the rich spots in various creeks, as if they followed the ancient outcrop of a gold-vein.

The depth of the laterite reaches 40, and even 60 m., according to the progress of decomposition effected during centuries by an immense amount of the richest vegetation in the world.

In one place, as described later, a quartz-vein always bearing gold was followed in depth for more than a hundred meters. But, in many other places, there was no downward continuation to be found, the veins being either too much disturbed, or irregular cracks in the rock itself, and not true fissure-veins.

The gold may be due also to segregation in volcanic rocks, like diorite, etc., as was the case with the mountain-dirt at St. Elie, in which the gold had been mechanically concentrated by erosion.

2. *Typical Details of the Upper Mana and Inini Placers.*

The government locations are of great size, generally more than 100 sq. km. The gold-fields in each location make various groups, each with a central place, or *établissement.* There are five or six *établissements* in a placer-location. While some creeks are regularly worked, the other creeks are actively prospected.

Work in each creek is started from the lower end of the gold-bearing ground, with a gang of eight or nine men and a foreman. They use a common sluice, easily removable, since the washing of gravel proceeds very fast in a creek-bed only 10 or 11 ft. wide. In advancing up-stream, the lower part of the sluice is removed to the upper end, when from the existing upper end the pay-streak is no longer accessible. In order to get a sufficient grade, the sluice starts from the gravel-surface, being supported by posts until it reaches the bed-rock downward. Its total length, rarely more than 20 m., is divided into adjustable boxes, each from 3 to 4 m. long. The daily progress is from 2 to 4 meters.

The pay-streak consists generally of small bluish or reddish quartz pebbles and white quartz sand, intermixed with a yellow or grayish clay from which the gold has to be separated in the sluice by mechanical agitation. This is women's work, yet not much more pleasant than shoveling in the tropical sun during the hot part of the day. The depth of gravel to the bed-rock in small creeks is rarely more than 4 or 5 ft. As I have said, many trees may have to be removed before the sluices reach their place. Men are employed to clear from wood, etc., the immediate continuation of the creek up-stream. If there are too many big boulders, that part is abandoned and the sluices are carried further up.

Sometimes, even in the dry season, the creeks are filled with water, which stops the work completely. In such cases, the Guiana people use a so-called *macaque* pump, which is a wooden balance-beam, supporting at one end a bucket, and at the other, a heavy stone as a counterweight. The water is thrown over behind a small dam, to prevênt its return into the workings.

I have seen many creeks exhausted for 1 or 1.5 km. in length, and their tributaries for from 600 to 800 m. The gold-

output from such work was sometimes from 150 to 200 kg., giving an average of 80 kg. per kilometer. But the picture is not always so bright. The width of gravel washed in one sluice does not exceed from 6 to 8 ft., and that may be sufficient in a very narrow creek, the more so as the miners try carefully to follow the richest channel. But often the creek widens, and it becomes necessary to put a second, and even a third, line of sluicing alongside the first. In rapid work this is frequently neglected, and a large amount of gold is left behind, especially if the rich channel has been missed. But, later on, when the creek is supposed to be exhausted, a party of unlicensed miners comes in from the old workings, and (even at a higher cost, since the tailings and barren gravel have doubled the over-burden of the pay-streak) digs again, and washes out gold. The distance from other creeks being great, nobody interferes. Such enterprises on the part of unlicensed miners have become an important feature of the industry.

A large amount of prospecting has to be done to keep an *établissement* in working-order, since the washing of a creek is rapid, and not every creek pays expenses. A year of work with a single sluicing-party represents from 1 to 2 km. of good-paying creeks, and twice as many prospected creeks. With three or four sluices, the proportion of prospecting is lower, since some creeks are wide enough for two or three lines of sluicing.

The prospectors have to take into account the thickness of the upper layer to be removed, the quantity of water, the amount of timber to be cleared out, etc. The prospects are usually trenches from 2 to 3 m. long, about 1 m. wide, and 10 m. apart along the creek.

As I have said, along the hills it is sometimes possible to find either some paying ground, or mountain-dirt, or heaps of quartz lying irregularly on the laterite. Such quartz has been found in some places quite impregnated with gold; it is then of a characteristic bluish color. I have seen many specimens of this quartz, but in no place has it been possible to trace it in depth or even follow it along what should have been ancient croppings, so much had the surface been disintegrated by erosion.

This quartz is in some places intermixed with the limonite,

or *roche à ravets*, suggesting the idea that the limonite might be the former iron-hat of the vein-outcrop, disintegrated like the vein itself.

The country-rock, judging from the fragments exposed in the worked-out creeks, is granite, or clay-shales, mica-schists, quartzites, white sandstone, even millstone, from which the Indians made their weapons. I have seen in the placers some of these weapons, well shaped and polished. They look exactly like the war-instruments of the stone age.

The creeks have been prospected for diamonds in certain places by screening the small pebbles. I made some trials of this kind, but failed to disclose anything like topaz, rutile, or corundum. I got only a few crystals of hornblende, tourmaline, and garnet, the usual minerals in granitic rocks. Yet it is in the same kind of creeks that the English Guiana prospectors succeed in procuring diamonds to the value of $50,000 yearly. It should be remembered that the proportion of diamonds in gravel is very small, and a pretty large quantity of gravel would have to be washed before any serious opinion could be given.

The gold-dust from the upper Mana placers contains from 96 to 98 per cent. of pure gold, while further below, and in many streams, it has only from 93 to 94 per cent. The last discovery (1908) in a tributary of the lower Mana river, at a placer named Kokinko, was an alluvial gold worth only 75 per cent. in pure gold. The production in a few months was about 800 kg., worth $400,000.

Supplies and Freight.—To avoid getting short of supplies in remote placers (from three to four weeks and more from the coast, by canoe), there are always to be found round the *établissements* some fields of corn, potatoes, sugar-cane, bananas, manioca, and the usual vegetables of a tropical, uninhabited country. In a well-managed placer-operation the use of preserved food is reduced to a minimum, yet it is unavoidable, as is also the presence of intoxicating liquors, always highly prized by the natives. At a few placers, I have seen some cattle, sheep, goats, and pigs, but these are luxuries, since they have to be brought in canoes. Chickens do very badly.

The great difficulty in these operations is that of providing the men with supplies. Every man needs (not including the

vegetables produced on the spot), say, from 1.5 to 2 kg. per day, or from 600 to 700 kg. per year. The 150 to 180 men not uncommonly employed on a placer-location thus consume from 150 to 200 tons a year: but, in view of inevitable losses along the river, and the proportion of damaged food, more than 250 tons must be provided. The freight is carried by the bushmen and Saramaca negroes, in their long canoes, each made of a single tree. From the last navigable place, or *degrad*, to the placer, the transportation is by portage on men's backs.

According to distance, the cost per ton varies from $100 to $200, but including the losses and damage, these figures may be doubled. The losses are due to upsetting the canoes in the rapids, but this is sometimes a pretext; as there is nobody to watch them, the men are easily inclined to steal food and waste their time along the shores of the river, shooting in the tropical forest, at their leisure and with intense satisfaction.

When rowing down the Mana river, I met some paddle-men doing their sixtieth day of canoeing from their starting-point, while others who had left at the same date had reached the same place in 22 days. Their delays had given time for the swelling of the river after rains, and the strength of the current had prevented any sensible advance by paddling.

Conditions in the Guiana placers have undergone a great change during the last few years. The procurement of supplies by the companies has become almost an impossibility. They have tried to lease their ground to the miners for a percentage of the gold (from 5 to 20 per cent., according to the conditions and richness of the field), and at the same time to get a profit from the sale of supplies. But the miners soon learned the advantage of doing their own transportation, and it is possible at present to see in certain places, either at a junction of creeks coming from various placers, or even in the middle of a placer, a native village with hundreds of packing-cases, and numerous warehouses selling all kinds of supplies, including, of course, liquors.

Labor.—The average number of miners and employees on a large placer-location with several producing creeks is as follows.

Mining,	75
Freight, transportation,	35
Temporary work, trails, repairs,	12
Stores along the rivers,	8
Canoeing, sick-list, etc.,	25
Total,	155

Among these there are from 15 to 20 women working in the creeks to clear the sluices from clay.

The above list does not include a certain number of men transporting supplies along the rivers, since these are usually paid by the ton, and the company sells the supplies, or at least a certain part of them, at as high a rate as possible, to make up for the losses on the river.

For 150 men, the monthly pay-roll is about $3,000, not including the supplies, which may double the expenses.

The wages per man for working in the creeks are $2.50 per day in the upper Mana placers; $2 and less in the lower Mana. But such high rates do not last long. They were reduced lately, on account of the smaller yield of the placers; and, as described below, many operators now work the creeks by the old method, buying the gold and not paying the men, but selling them the supplies.

Returns.—The Souvenir placer was one of the last to give high returns in gold. Along 12 to 13 km. of creeks (not exhausted) the production was:

	Kilograms.
1898 (six months),	72.527
1899,	183.484
1900,	138.247
1901,	127.935
1902,	120.170
1903,	319.571

After 1903 the production decreased to about 100 kg. in 1908. The total gold produced by Souvenir in less than 10 years has been more than 1,500 kilograms.

Average Product of Gold.—In the great placers of the upper Mana river, the gold-product, per kilometer of length along the creeks, has been from 50 to 80 kg. This is about the same result as that of the famous St. Elie placer, the product of which has been 6,000 kg., or an average of 60 kg. of gold per kilometer for about 100 km. of ground sluiced.

The daily progress in drifting is from 2 to 4 m., but, including days of rest and repairs, it cannot be estimated at more than 2 m., or from 50 to 60 m. a month, or 600 m. a year. The gross product in the placers during my visit was from 60 to 120 g. per day and per drift (or 2 m. progress).

The average width of a creek is from 4 to 5 m., and the thickness of the pay-seam 0.30 m. The total ground removed in a day is thus from 5 to 6 tons, averaging (including the barren ground) from 2 to 3 g. of gold per ton, or about 5 g. per cubic meter. The cost per cubic meter may be estimated, in the upper Mana, at about 3 g. of gold. In the lower rivers, the cost is lower, and it is thus possible to drift in creeks that would not pay in the upper rivers.

Unlicensed Miners.—The class called *maraudeurs* are nothing else than unlicensed miners. To get a right to work in Guiana, it is not only required to have a miner's license, but to make application for a government survey, and have the limits of location marked upon the ground and across the forest. This is a very onerous operation in an uninhabited tropical country, and often a long way from the sea-shore. Consequently, many adventurers dispense with such formalities and become *maraudeurs*.

A few years ago, in order to get rid of these men, it became necessary to employ police and troops. In a remote placer of the Mana, shortly before my visit, there had been a rush of 250 or 300 unlicensed miners, who came along the Maroni river. The mine-owner was forced to organize, at his own expense, a regular army with a hundred soldiers, their officers, a doctor, and their trains of supplies. This condition lasted more than two months, but the success was complete, and there was not a single man of his army taken sick, or drowned in the cataracts and swift currents of the Mana.

Not all the doings of the unlicensed miners are to be condemned. They have a right to prospect, and perhaps their discoveries should be better protected. They look for regions of doubtful nationality, entirely free from police, justice and custom-taxes; and thus, in a certain way, they go ahead of civilization. It was in this way that they rushed to the Awa placers, along the upper Maroni, between the French and Dutch Guianas; to the Carsewene, in contested country be-

tween Brazil and French Guiana; later, to the Inini river, not far from the boundary between French and Dutch Guiana. Their last discovery, 1907, was the Kokioko placer, on a left tributary of the Mana, where they got about 800 kg. in seven or eight months. The actual production of Kokioko is more than 2,000 kg., but the rich pay-streak has sunk below the hills and its exploitation now involves expenditures beyond the means of unlicensed miners.

For the present moment, as the rich discoveries seem to be at an end, except perhaps in the hinterland, the companies have begun another policy by accepting readily the unlicensed miners in their placers, letting them work at their convenience, and buying the gold from them. At the same time, they try to control the market for supplies, and get the benefit of high prices, so long as they can. But the manager of a placer is not always a wise man; and it happens that the miners, discontented, get their own supplies, or that another class of men, always on the watch for an opportunity, bring in goods, build stores, and enter into fierce competition with the company stores. I have seen lately such villages of miners, with a hundred stores and more (of course these stores are nothing but small straw huts), along the Mana and its tributaries, and also along the Approuague (east of Cayenne). They give some picturesque names to these places, like Delices, Pays, La Louise, etc.; and their bustling activity is in striking contrast with the surrounding silent forest.

III. Dredging in the Rivers.

So far, it has been possible to get gold in French Guiana without employing improved industrial methods or working on a large scale. The output of small creeks gave a sufficient remuneration, without any large outlay of capital. But when during several years the yield of gold gradually diminished, it became impossible to continue to pay high wages, or to spend the large amount necessary to exclude or expel unlicensed miners from the locations of companies. Lately, as I have said, there has been an understanding between companies and unlicensed miners, to let the latter work on the sole condition that the gold be sold to the company at a fixed price. But the quantity of gold stolen, and the competition between

the miners' and the companies' supply-stores, have so much reduced profits that the promoters of new companies must look for a better industrial system.

Hydraulicking has been tried in British Guiana, with pumps to elevate the water; but the high cost makes it impracticable. Consequently, dredges and excavators remain as the only alternatives.

For many reasons, among which was the necessity of additional manual labor, the excavator has not been tried. Dredging has been introduced during the last two years, in three places: the placer Elysée, on a tributary of the Mana river; Sparwin creek, along the Maroni; and the Courcibo, a tributary of the Sinnamary.

1. *Advantages of the Method.*

The main advantage of dredging is its low cost. If the field is not very extensive, a plant of 1,000 cu. m. daily capacity may suffice, and may cost $60,000, including its transport and erection on the spot. Owing to the excessive wear and tear in tropical climates, the amortization should be done in six years, at, say, $10,000 a year.

Supposing that, by reason of the numerous obstacles in the creeks and in hauling the gravel, the real efficiency is lowered to 600 cu. m. a day, with 300 days of regular work in a year, then the amount of gravel washed will be 180,000 cu. m., and 5.5 cents per cubic meter will be required to repay the cost of the plant in 6 years. If the dredge can work 10 years, this will be reduced to 3 cents.

The working-costs in Guiana are very high; but they probably will be reduced after a few years of experience. For the present, it is difficult to reckon with less than from 30 to 40 cents per cubic meter.

A great advantage of dredging is that prospecting-work may be executed with great accuracy. It is easy to dig out the ground along the sides of the river; and this ought to be done in Guiana, since the proportion of gold is very often as high under the bank as in the bed. It should be borne in mind, however, that the rivers suitable for dredging under present conditions are not the many smaller streams, but the big creeks, unsuitable for sluicing, yet not very wide, the water-

course not exceeding, and sometimes not equaling, 10 m. in width.

To test the ground, the diggings are made large enough to allow the extraction of 1 or 2 cu. m. of the pay-streak. This gravel is washed in a long-tom, giving thus an accurate result. Such diggings are made on both sides of the river, 30 or 35 m. apart. The cost for 2 km. of prospecting is about $6,000, but it must be done before erecting a dredge that may cost ten times as much, and give bad results.

The irregularity of gravel makes it advisable to dredge first the rich spots, thus gaining experience, and to dredge afterwards, with more economy, on the low-grade places which might cause a loss at the start.

2. *Conditions of Success.*

For the success of a dredge, the following conditions may be laid down as a rule:

1. Close prospecting to expose the paying ground, and secure at least five or six years of dredging.
2. Soft bed-rock, easily taken up by the buckets. This is the material richest in alluvial gold.
3. Absence of large blocks or boulders. The tree-stems, etc., may be removed without too much trouble.
4. Absence of cemented gravel. If the gravel is mixed with clay, the dredge will have to be provided with an apparatus to disintegrate it.
5. A dredging-depth of not more than 10 m. It is exceptional for dredges in California to dig as deep as 15 or 20 m. The depth is the main factor to be considered before ordering the dredge, with its gears, pins, and various appliances.

3. *Costs of Dredging.*

The following is a complete estimate of costs at the placer Elysée, after experiments lasting about six months with a medium-sized dredge of English design (Renfrew, Scotland). While this dredge, working at full capacity, should have washed 1,000 cu. m. in 24 hr., it was impossible, for reasons stated later, to wash more than from 300 to 400 cu. m. in 24 hr., during the first season.

	Per Day.
Fuel; 23 cords at $1.30,	$30.00
Labor; 10 men per shift (8 hr.) at $1.12 (including food),	34.00
Engineer and foreman,	5.50
Clearing the wood and sundries,	16.00
Repairs, water, tools, etc.,	5.00
Freight, general expenses, management, etc. (subject to change),	40.00
Total costs per day,	$120.50

In each shift there are 10 men: one winchman, one engineer, one fireman, one man and a helper for washing, three men with shovels for cleaning the gravel from adhesive clay in the buckets, three men for transporting the wood, tying the ropes, etc. The wages average $0.80 per day, and $0.32 for the food, according to contract.

In 1907–1908, two dredges on the Lezard creek, operating with many interruptions, gave the following results with the above costs:

	Gravel. Cubic Meters.	Gold. Kilograms.
No. 1,	126,000	80
No. 2,	40,000	18

In Siberia it has been possible to work easily with a monthly cost of $1,600 to $1,800, while in Guiana it costs as much as $3,600 per month, and 36 to 42 cents per cubic meter, washing only from 300 to 400 cu. m. per day. As will be shown later, these conditions are likely to improve a good deal upon further experience.

4. *Difficulties of Dredging in Guiana.*

Present local conditions and difficulties cause expenses to be much higher in French Guiana than in other countries.

First, there are the enormous trees of the tropical forest, on both sides of the creek, which must be pulled down and burned. The creek itself is seldom more than 8 or 10 m. wide; but dredging should be done over from 30 to 50 m., so as to cover the beaches and side-swamps, which are also gold-bearing. The Guiana rivers are subject to many windings; and the gold exists as well below the sides as in the river-bed proper.

Pulling down the trees is tolerably easy, because the roots spread themselves almost on the surface, so that the fall of a tree carries away its neighbors. But burning these hard woods after their fall is a more difficult matter.

A second drawback is the great quantity of dead wood, trunks and branches entangled together in the gravel and the gold-bearing clay-dirt. A small-sized dredge may be unfit to drag them out; the back part rises while the fore part descends and may sink in the water. A more powerful dredge, with a skillful winchman, does better.

Fortunately, no big boulders exist, as a general rule, in the dredgeable creeks, although they have been abundant in the small creeks washed by hand-labor.

The bed-rock is quite soft, letting the buckets penetrate it easily, thus facilitating the extraction of gold from the richest part of the deposit.

The beaches are often 3 or 4 m. high above the pay-dirt, which has an average thickness of from 30 to 60 cm., and is composed of vegetable soil, clay, sand, and gravel, with decomposed rock. The proportion of barren ground to be washed is thus from six to seven times the pay-dirt. In California and elsewhere, the upper layers often contain gold through the whole thickness of from 5 to 10 m.; but in Guiana the bottom alone is rich. It would seem advisable to erect a special apparatus to remove the barren ground; but this would double the expense, and the ground thus removed would be cumbersome for the dredge behind.

I have already mentioned a very tight clay around the pebbles, which becomes a nuisance in the sluice, in the shape of clay balls, the gold inside of which cannot be washed out without a complete cleaning, performed by women standing in the sluice. In a dredge, I have seen this clay so tightly stuck to the buckets that it came again and again at every turn, despite all water-jets, and it was necessary to have three or four men with shovels standing on the platform to separate the clay from the edge and bottom of the buckets. A mechanical apparatus should be devised for that purpose.

It is almost needless to say that the slimy ground intermixed with rotten trunks and boughs, often of malodorous wood, is a perfect field of fever. One must be very healthy to resist its effect, especially as the climate itself produces anæmia and fever. The dredge has to operate in a regular swamp, subjecting the men to all the dangers of paludism.

The natives are very awkward workmen. At some mines,

like Adieu-Vat, the experiment of importing a few Italian miners has been tried; but it resulted in loss. The native is used to the food and other climatic conditions, while a European needs better food; and these Italians, no better used than the natives, either died or had to return to Europe. This condition helps to explain the fact that, although for more than 50 years rich deposits of gold have been known to exist in French Guiana, they have never yet been systematically worked. It is only just to add that the placers exploited by the crude methods above described have paid large profits without such organization. For the work of dredging, they have succeeded, at Elysée, after much trouble and the exercise of a good deal of patience, in putting the natives in charge of a dredge, on the condition of having a white man as overseer to be ready for all emergencies.

There is another and final difficulty in Guiana: the transportation along the rivers of the heavy pieces of machinery. The only vessels are canoes, and the rivers are subject to frequent and sharp turns. A certain dredge failed to reach its destination, because the hull-pieces were too large to be got over the first rapids. They are still to be seen along the trail of the portage. The attempt was made to remedy this loss by building a wooden hull, at the placer; but in that tropical climate insects and worms eat away the wood in a short time, and when another hull, ordered in smaller pieces, had arrived, the upper part of the dredge was badly damaged also.

Obviously, such accidents and the difficulty of repairs are serious drawbacks to dredging-operations in a remote place. They may stop production for many weeks, while expenses are running on. An experienced, prudent and resourceful superintendent can prevent or avoid much trouble. This is a factor in the problem of success that cannot be put into figures. In Siberia, the superintendents are mainly Americans and New Zealanders.

5. *Prospecting.*

Proper prospecting is the main element of success in dredging. It is an easy operation in Guiana. An area of 4 or 5 sq. m. of ground is excavated as far as the pay-streak; a wooden frame, about 2 m. long and 1 m. wide, is used to prevent the

walls falling in, while the gold-bearing gravel of that layer is being shoveled to the surface. Washing is done in a long-tom, and the gold is weighed. With about 60 diggings of this kind along 1,500 to 2,000 m. of a creek, the value of ground may be fairly estimated. Experience having shown that about 40 cents per cubic meter is necessary to cover all expenses, the prospecting should give returns higher than that, before the expense of a dredge is incurred.

6. *Labor.*

Neither the home nor the colonial government has done much in favor of industrial undertakings. A striking contrast is presented in Dutch Guiana, where a 150-km. railroad is building towards the Awa, with a branch to the Albina harbor, while the Dutch government is strongly supporting agriculture, and more specially the cultivation of the banana and other fruits, which are transported in large steamers to New York and to Europe. British Guiana is still more prosperous. In that colony, dredges are successfully operated upon gravels much inferior in gold-content to those of French Guiana.

It might seem that better labor could be obtained in French Guiana by employing the convicts. So far, the results of such experiments have been far from encouraging. Yet another is to be made on a placer not very far from S. Jean du Maroni, the "relegate-colony," the outcome of which will be interesting. The inherent laziness of that class is little affected by rewards or punishments, unless adminstered by overseers of special tact and influence. On the other hand, working on a dredge is not so hard for a white man as digging in a placer, and perhaps dredging might become a moral benefit to these convicts, as agriculture and cattle-raising became to the same class in Australia.

7. *Experiments and General Results in Dredging.*

Dredging has been tried, so far, on three rivers only: the Curcibo, the Sparwin, and the Lezard. Three new dredges are in course of erection in those three places—an indication that the first results, either in dredging or in prospecting, have been encouraging. There are in French Guiana other rivers equally rich. According to the latest reports, a new dredge,

operating in Roche creek, carefully prospected in 1908, has given an output of 8 kg. for March, and 10 kg. for April, 1910.

The principal problem has been to settle beyond doubt the mechanical practicability of dredging; its necessity being in that case clear, since all the small creeks, suitable for sluicing, are nearly exhausted.

The general results obtained are the following:

1. There is no more difficulty in removing the trees and stems in the gravel than in removing the quartz boulders; it is a matter of practice.

2. A good deal of clay can be well washed in the trommels by means of water-spouts. For the more compact clay sticking to the buckets, a mechanical way of cleaning should be devised.

3. It is possible to educate a native shift to work on a dredge. This is an important result, since for 50 years the labor-difficulty has been the greatest in all undertakings in French Guiana. A white man as overseer, however, is still a necessity.

There is much trouble at present, because of the new conditions. The old sluicing-process that gave full satisfaction is to be abandoned, in favor of a new process that has to fight against adverse conditions; the alluvial ground of Guiana being entirely different from the gravels in California and elsewhere.

A dredge for work in Guiana should be, in my opinion, provided with the following modifications:

1. An automatic cleaning of buckets on the top of gear.

2. Buckets of a rather flat shape for easy cleaning.

3. An elevator with an endless belt; or rather no elevator, but the sluices running some length backward from the dredge.

4. A direct transport of the barren gravel to the rear of the dredge, to avoid obstruction.

5. A special device for getting pure water into the pumps.

6. A riffle-table for saving the nuggets.

Of course, there is much irregularity along the pay-streak here as everywhere. Along the windings, it would be expected, as was the case in famous placers of California, that a concentration of gold should have taken place. But this has

not been observed in Guiana, possibly because the current is slow, and the windings are, perhaps, of recent origin. In fact, the rich pay-streak sometimes cuts across the windings below the barren ground, as if there had been an older channel for the stream. (But this has nothing in common with the "old channels" of California.) Thus dredging should rather follow the line of richness as indicated by the prospect-diggings.

As a rule, the gold in the main creeks comes from the tributaries; seldom direct from quartz-veins in the bed-rock. It is in small particles, pretty evenly distributed over a certain length.

Undoubtedly it is easy, through lack of prudent preliminary investigations and calculations, to waste money in dredging-operations. Experience has shown, even in New Zealand, the mother-country of dredging, that too often the capital expended is greater than not only the dividends afterwards distributed, but also the gross product of gold. This might happen in French Guiana, if not sufficient care were taken in prospecting.

As I have said, the small creeks are becoming exhausted, and there are no new discoveries. The unlicensed miners themselves are working the main creeks, when there is not much water, and where the banks are not high. It is much more convenient to dredge in water and high beaches.

It may be that the main creeks contain more gold than the small ones. The problem is to get it with less cost. The total gold-product of French Guiana for 50 years, including everything, is certainly less than $100,000,000. This is not to be compared with the other alluvial gold-fields of the world, like California, Australia, and Siberia. Even Alaska has yielded $100,000,000 in ten years.

8. *Commercial.*

In French Guiana, as in Siberia and other countries, where placers are far from towns and villages, the mining companies are wont to keep stores of supplies of every kind, and to make a profit upon their sales to the workmen. It may happen even that the main profit of the undertaking comes from the sales, when the miners and employees are fairly numerous, as in Siberia.

According to their general contract, in French Guiana, the workmen have a right to rations from the employer, as a part of their wages, but these rations are fixed at a minimum, and they buy the balance from the company's stores. The following is the contract-food of the miners, besides their money-wages of from \$0.70 to \$1.10 per day.

	Per Week Grams.
Salted meat, lard, or codfish,	1,500
Beans or dried vegetables,	800
Manioca, rice, or bread,	5,000
Oil,	250
Tafra (rum of second quality),	40
Tobacco,	30
Total,	7,620

The total cost of these articles at the sea-shore is \$1.53, but including the freight (from \$60 to \$200 a ton) from the coast to the placers, the cost at the placer is from \$2 to \$3 or more.

The meals of the miners are not much superior to those of convicts, except that they get a little more lard or codfish. But they have pretty hard work, while the convicts have little to do. Yet the death-rate of convicts is 12 per cent. in a year. The life of a Guiana miner, with the ordinary rations, has little comfort in it.

The companies' stores may sometimes increase their profits by selling supplies to unlicensed miners, but these have usually soon discovered the advantage of combining to run their own stores; practically, the trade of a company is almost entirely confined to its miners and employees.

IV. The Quartz-Ledges.

Although dredging has nothing to do with quartz-ledges, a few words about the quartz-discoveries in French Guiana may be of interest.

As remarked in the above description of the placers, there is a good deal of more or less auriferous quartz along the creeks, but in almost every one of the diggings no quartz has been discovered in depth. The depth of laterite, or decomposed rock, being often 40 or 50 m. or more, it is evident that the search for a ledge might be very expensive. These surface-pieces of quartz, sometimes very rich, may be seen at many

places; the upper Marna placers, St. Elie, Elysée, Adieu-Vat, etc. Indeed, a regular quarry was opened at St. Elie, and quartz was crushed in a 3-stamp battery during 1901 and 1902. For 1,530 tons crushed, the output was 72.306 kg. of gold, or an average of about 50 g. ($30) a ton.

The same kind of work was done later at Adieu-Vat. A quarry (in soft laterite), 200 m. long and 50 m. wide, was opened across a hill. The quantity of quartz taken out was about 12,000 tons, one-half of which is still on the ground. The other half was crushed, and the red dirt containing gold was also washed in a sluice. The result was somewhat inferior to that at St. Elie.

But during operations at Adieu-Vat, a quartz-ledge was at last discovered in the compact rock, more than 200 m. long. The country-rock is a greenish or blackish diorite. The vein is 1 m. wide, but one-fourth only is mineralized; it dips 70° N. The gold was first mixed with pyrite, but further down with tellurides and bismuth sulphide. The quartz is bluish and unctuous.

During 1903, with a small 3-stamp battery, 419 tons of quartz crushed gave an output of 61.650 kg., or 145 g. per ton. The value was $36,475, or $86 a ton. But the value of quartz has been decreasing rapidly in depth, even taking into account that a much larger proportion of rock was crushed, without sorting, in a 13-stamp battery. The proportion of gold saved was only 20 or 25 g. per ton, but probably a similar proportion was left in the slimes to be treated later on by the cyanide-process.

The vein was developed along five levels at the respective depths of 20 m., 34 m., 52 m., 85 m., and 115 m. (or almost 80 m. below the bed of the Sinnamary river). The length of the levels varies from 80 to 289 m. The gold seems to follow an ore-shoot extending irregularly in the various levels.

Two more rows of surface-quartz were tested in depth with inclined shafts and tunnels, but failed to prove really fissure-veins. It may be said, therefore, that, for the present, the Adieu-Vat vein is unique in French Guiana.

Mining in Nicaragua.

BY T. LANE CARTER, CHICAGO, ILL.

(Canal Zone Meeting, October, 1910.)

Introduction.

It is a curious fact that while in our *Transactions* there are papers dealing with mining-districts in all parts of the world, in Europe, Asia, Africa, and Australia, there is not one which describes the mining possibilities of Nicaragua, our near neighbor. To be most interested in distant objects, like the moon, and to neglect what lies at our feet, appears to be a trait of human nature.

The neglect of Nicaragua is not confined to the American Institute of Mining Engineers. In no other transactions am I able to find papers dealing with mining in Nicaragua, some parts of which are as much in need of discovery and research as the North Pole. I hope this paper will be the precursor of many others to be printed in our *Transactions*, dealing not only with the mining possibilities of Nicaragua, but with the whole of Central America, where there are vast areas awaiting skill and capital for their development. Much of this capital and skill should come from the United States in the future, although it is probable that European capitalists, when they appreciate fully the possibilities, will investigate more and more the mining-resources of Nicaragua and other Central American republics. Fig. 1 is a sketch-map of Nicaragua showing the principal mining-districts.

II. History.

Mining in Nicaragua did not begin yesterday. It was commenced by civilized man when the Spanish conquerors penetrated into the interior and forced the natives to procure the precious metal for them. It is strange that the Spaniards did not find more gold, and work on a scale as extensive as in Colombia and other parts of South America. Probably in Nicaragua the Indians covered up most of the rich prospects

to hide them from the cruel and fanatical conquerors, and as the Spanish were not prospectors, they spent little time looking for gold.

On the Atlantic side of Nicaragua, the Spaniards did not explore a great deal, and so this promising section of the New

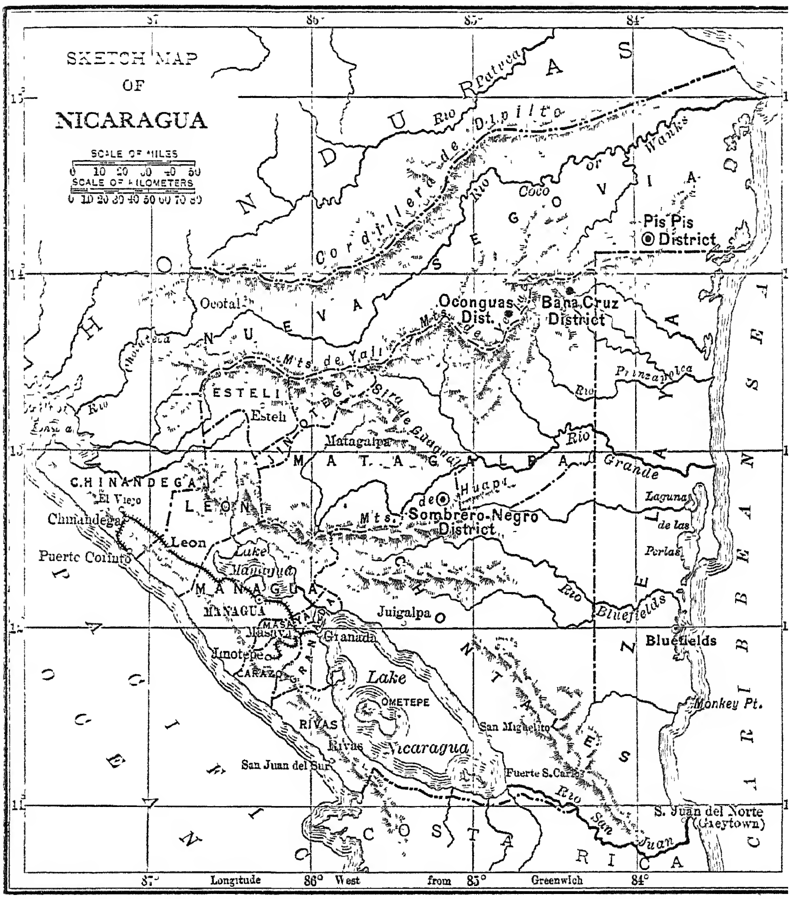


FIG. 1.—SKETCH-MAP OF NICARAGUA, SHOWING LOCATION OF MINING-DISTRICTS.

World returned them very little gold. Until recent years this part of Nicaragua remained unknown.

During the gold excitement in California, say from 1850 to 1860, mining in Nicaragua took on a new lease of life, for some of the adventurers were so attracted by the appearance of the country on their way to and from California, that they

remained in Nicaragua. Numerous small mines were opened in Leon, Matagalpa, and Segovia. Most of these enterprises were short-lived; but a few survive. The Santa Francesca, 10 miles from Leon, and the Santa Rosa, are still worked, while the Leonesa, in Matagalpa, promises to become important.

Few of the workers of that period have left us their experiences; but one man, in a charming book[1] now out of print, has given a vivid account of the difficulties faced by the mine-manager in Nicaragua. It was before the days of the cyanide-process, and he complains of his low "extraction." Had the metallurgy of gold been known then as it is known to-day, the subsequent story of Nicaragua would have been different.

In the Liberated district, where conditions are favorable, numerous small properties have been producing gold for years, and many Nicaraguans, in spite of all drawbacks, chief of which was "high-grading" by their employees, have realized from the mines of this district what to them were fortunes. Modern mining is almost unknown there, the work being carried on with the crudest machinery, by people not acquainted with the art.

About 1888 the first discoveries of gold were made in eastern Nicaragua, and on the Atlantic coast. The placers did not last long and were soon abandoned for vein-mining. No attempt on a working scale has been made at dredging, the indications not being favorable. A promising field for investigation is furnished by the black sands on the coast, especially from Walpasixa along Cape Gracias. But the future of gold-mining in Nicaragua depends on the development of the veins.

The mining on the Atlantic coast has been done by men with little experience or capital. A man, probably "grub-staked" by some merchant in Bluefields, would go out into the wilderness with inadequate equipment. Too often the merchant has looked upon mining as a method for making a market for his wares. If the small mining company which he formed bought a sufficient quantity of his goods, and was able to pay his bills, he was satisfied. In one instance known to me, a promising mine was almost ruined, because several mercantile firms were interested in it, and each had the right to

[1] *The Naturalist in Nicaragua*, by Thomas Belt (1874).

supply goods for four months in the year. During these periods the firms would unload the maximum amount of supplies on the mine, each firm, in spite of the manager's protests, sending him in four months provisions enough to last him a year. The consequence was a criminal waste. It was not uncommon to see barrels of flour, costing $25 each, strewn along the wayside. Under such conditions it is no wonder that the mine did not pay dividends.

Some investors in gold-mines declare that they want a mine that will succeed in spite of bad management. In other words, the mine must be so rich that it will almost run itself. Such people should keep away from Nicaragua. It is not a country of bonanzas. Rich pockets have been found, but they are rare; on the whole, the region requires most careful and capable mine-management.

Too much of the failure of mining in Nicaragua in the past has been charged to the government. The administration of Nicaragua has been, indeed, responsible for the lack of confidence on the part of outside capitalists in the stability and safety of the country; but it cannot be justly blamed for all the numerous disasters due to the ignorance and incompetency of the operators themselves. The greatest harm done by ex-President Zelaya to the mining industry of Nicaragua was to frighten capital away. The few men who have had the courage to go into mining there, have found that, given a paying mine and capable management, they could make money in Nicaragua, even as was done in the Transvaal during the Kruger *régime*, which was certainly unsympathetic to the mineral industry. So capable was the management in South Africa, however, that great success was attained there in spite of Kruger.

The granting of concessions by Zelaya to companies or individuals did more to prevent the exploration of the mineral wealth of Nicaragua than any other cause. In traveling through the country, the mining engineer, after seeing many promising prospects, wonders why the field is not more vigorously explored, in view of indications which, in the United States, would set hundreds of prospectors at work. The reason is, that the policy of Zelaya prevented the general prospecting of the country. If a bold explorer went into a new, wild district and located a mine there according to law, he would have

no trouble in getting clear titles to his particular location; but if he started work and made a success, the chances were that Zelaya would grant a concession of all the surrounding country to some company in which he was a large owner. The worst of it was that after the concession had been granted, nothing further was done except to exclude prospectors, and promising tracts of mineral land have remained untouched for years because of the Zelaya policy. The worst example of wasted money and resources is that of an American company to which Zelaya gave a concession covering a large tract. Incompetent and dishonest men squandered in Nicaragua thousands of dollars of the company's funds, spending money on boats, hotels, etc., but none on the mines. The only satisfaction that the company has to-day is the fact that no one else has been able to open gold-mines on the land granted to it.

In my journeys through different parts of Nicaragua, I frequently asked the natives if they knew of any promising prospect. "Yes, sir," might be the reply. "Over there on the mountain-side we have found a vein that is wide and rich."

"And why do you not prospect and open it up?"

"Because it is on the concession. If we work hard and prove there is a mine on that mountain, the men who own the concession will come and run us off. So we cover up our pits; and by and by, when Zelaya and his concessions pass away, we will work there and show what we have."

Not only are such mineral concessions contrary to the Constitution, but all of them contain loop-holes, so that they can be annulled under some pretext or other. These concessions have often been obtained by parties who had not the slightest intention of working the properties, but planned to sell them to the uninformed foreigner for whatever price could be obtained. The popular feeling is bitter against concessions, and now that Zelaya has been deposed, concessions will be abolished, and the country thrown open to prospectors. If this be done, I look for a period of brisk exploration in sections where the indications are favorable.

There has been a feeling in the outside world that mining-property is unsafe in Nicaragua; that, even if a company or an individual owns a mine, there is constant danger of spoliation or confiscation. My experience has proved that this fear is

unfounded. If a foreigner takes pains to see that he has complied with every law—and of this he must be quite sure—and obtains a clear title to his property, has it registered according to law both in Nicaragua and in his native country, he is as safe in his possession as if the mine were in England or the United States instead of the wilds of Nicaragua. The trouble in the past has been that the foreigners have not been sufficiently careful in complying with all the laws. As long as a company or an individual has a clear case and can appeal to his government, he will always get justice, especially if he be an Englishman. While high-handed methods are sometimes used with the natives, these tactics are not employed with foreigners.

III. Mining-Law.

The mining-law of Nicaragua is one of the most liberal in the world, especially from the poor man's point of view. The taxes required by the government on mining-claims (*pertenencias*) are only $5, in Nicaraguan money, about $2 in U. S. currency, per hectare per year. A hectare is 14,184 sq. yards.

The mining-law is liberal in its granting of sites for plants (*planteles*) for the treatment of ores and also for water-power. The government allows a maximum of 100 hectares for a mill-site (*plantel*), charging only 80 cents, U. S. currency, per hectare per annum.

The law requires prompt payment of these taxes before the first of February in each year. Otherwise, the property is put up at auction. The mining-claims only lapse by failure to pay the license within the specified time. No assessment-work of any kind is required, and by the payment of the small tax the owner can hold his claims indefinitely. Few countries in the world offer such easy conditions.

The shape of the claim in Nicaragua varies with the dip of the vein. The width of the claim is determined by the dip, according to the following table:

From 30° up to 45°, 200 m. wide.
From 45° up to 50°, 165 m. wide.
From 50° up to 60°, 135 m. wide.
From 60° up to 65°, 115 m. wide.
From 65° up to 90°, 100 m. wide.

After the width of the claim is decided on, the length of the claim is figured out, using the width as one side of the rectangle, to find the other side. Fortunately for the country, the boundaries are vertical planes, as in the Transvaal, so there is no complication or litigation over extralateral rights, etc.

The discoverer of a new mining-zone is entitled to three claims; all others can acquire only one claim each. This limitation of the amount of ground that a man can take up in a district would be very awkward, were it not so easy to get round the difficulty. If a merchant desires ten claims in a recently-discovered mining-district, he stakes out one claim in his own name, and nine in the names of his clerks. Later on, when the titles have been granted, the clerks transfer their claims to him.

Every facility is offered for the acquisition of title. The judge of the district appoints a competent man to measure the claim, the owner of course paying for the surveying, stamps, etc. It is required of owners that they keep the monuments of their claims in order, and also keep the boundary-lines cleared. The miner is allowed to use for fuel or lumber the wood in the surrounding forest.

Provisions are made for the safety of the miners as regards ventilation, timbering, etc.

One provision strikes the engineer as peculiar. "In the preparation of the shots (that is to say, the ramming of the powder in the drilled holes) only ramrods, the outside surfaces of which are of soft iron, bronze or other metals which do not produce sparks, shall be used." If this provision be conscientiously obeyed long enough, the miner should eventually blow himself up!

The employer is required to provide for the treatment of his laborers in case of sickness or accident. Supplies and mining-machinery of all kinds are admitted free of duty. There is a direct tax amounting to about 5 per cent. on the gross output of gold. Indirectly, the burden of taxation on the mines is great, on account of the heavy duties levied on food-stuffs.

The acquisition of water-power for mines is easily accomplished by simple denouncement.

IV. River-Transportation.

The greatest drawback to the development of the country is the difficulty and cost of transportation. One of the first things a large corporation should do in entering Nicaragua is to build a railroad either from the coast or from a point on a large river to which boats could travel all the year round. While the country is not an easy one through which to build a railroad, the difficulties are comparatively small, save in crossing rivers and creeks, where provision must be made for the sudden flooding of the streams.

In the Pis Pis district, one is amazed at what has been accomplished in spite of the difficulties of transportation. Were it not for the numerous streams, which enable the miners to bring freight near their doors, such mines as the Bonanza, Mars, Lone Star, Siempre Viva, and Constantia, could never have commenced operations. All the machinery and supplies for these mines were paddled up the sinuous streams by the Indians in the same kind of canoes (called "pit-pans") that Columbus saw centuries ago. Figs. 2 and 3 show these pit-pans. One cannot help admiring the marvelous skill of the Indians in the canoes. They hop and yell, in starting out, from the joy of the work, and if you have a scalp-lock, it trembles! But red men have paddled me day and night for nearly a week, with only snatches of sleep, displaying an endurance that I never suspected, and a skill which is inherited from generations who have practically lived and died in these canoes.

There are occasional diversions on the long canoe-trips up these rivers. Now and then the Indians draw up on a bank of sand to get turtle eggs, or one of them will plunge after an iguana, the huge lizard, which is considered a great delicacy by the Indians. Creeping up on alligators which are sunning themselves, is also a frequent pastime.

The newcomer looks aghast when the Indians begin to push the craft foot by foot up some tumbling rapids which seem insurmountable. But, after having learned how expert they are in such troubled waters, he gains confidence, and remains in the canoe, even when he learns there is a water-fall ahead. Considering the thousands of tons of machinery and supplies,

mortar-boxes, stems, boilers, Huntington mills, etc., that have been paddled up these streams, one is surprised that so little has been lost by wreck in transit. The credit is not due to the natives only, but to the supervision of the merchants at the coast as well. By far the greater part of this freighting on the rivers in pit-pans is done by Silverstein & Kelting, of Prinzapolca, who handle the freight on contract, at so much per pound. Every canoe which leaves the large warehouse at Prinzapolca is inspected by Silverstein & Kelting, and sent up the river in charge of an Indian captain.

The scheme of using gasoline-boats instead of these canoes is one that suggests itself to the engineer at once. The idea has been tried: but there are several reasons why the Indians still handle the freight. In the first place, it would be almost impossible for a gasoline-boat to get over the rapids in the rivers. Again, it pays the merchants to use Indians, for they get back the wages of the natives by selling merchandise to them. Nevertheless, I believe that gasoline-boats could be made a success, and I think they will be introduced before long.

The distance made per day in paddling up the river varies with the season. In the wet season, from May to December, it takes twice as long to go from Prinzapolca to Tunky as in the dry season. Often when the floods come, the rivers overflow their banks, and one can travel down stream at a terrific speed, making almost a straight line for the coast, and paying little attention to the river-bed. To come from the interior to the ocean in flood time is as exciting a race as a man could desire. It doesn't take long to get there, even if he doesn't arrive alive.

V. Land-Transportation.

Nicaragua is an exceedingly wet country. The amount of moisture in the atmosphere is excessive, and the effect on such things as books (which mildew rapidly), assay-balances, surveying-instruments, etc., gives the resident much annoyance. At Greytown, at the mouth of the San Juan river, on the coast, it rains most of the time, the rainfall some years amounting to 300 in., while at Bluefields, to the north, the rainfall is as high as 180 in. per annum. In the Pis Pis mining-district it averages 125 in.

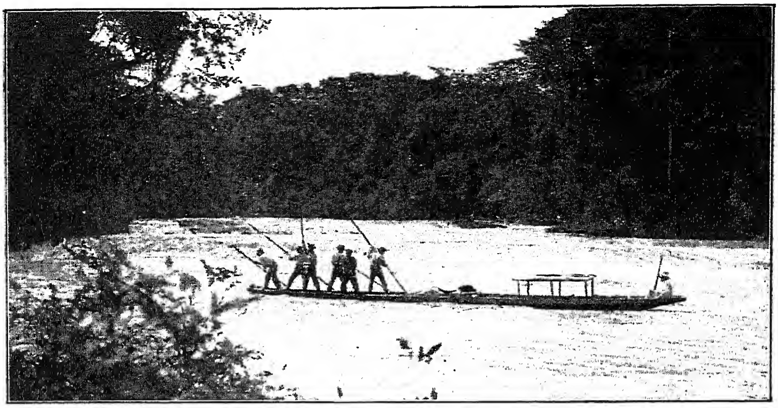


FIG. 2.—PASSENGER CONVEYANCE ON PRINZAPOLCA RIVER.

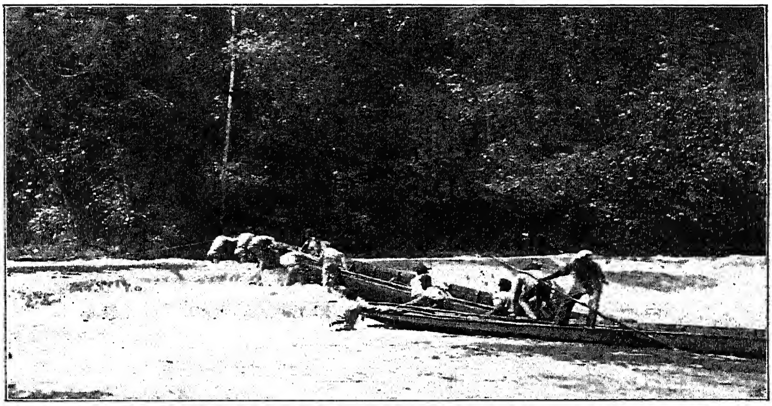


FIG. 3.—PIT-PANS GOING OVER FALLS ON PRINZAPOLCA RIVER.

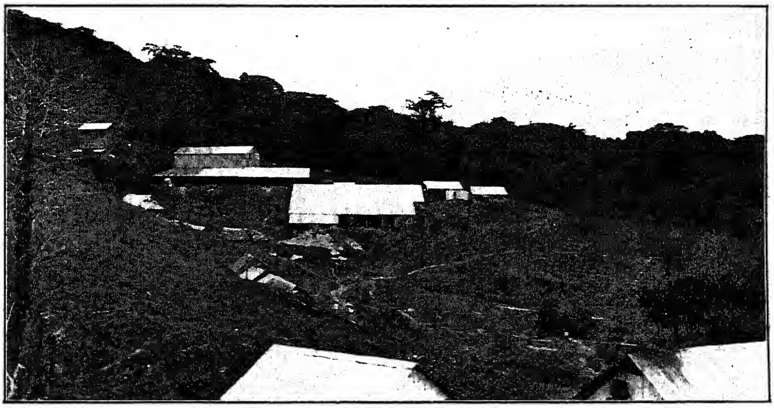


FIG. 4.—LONE STAR MINE, PIS PIS DISTRICT.

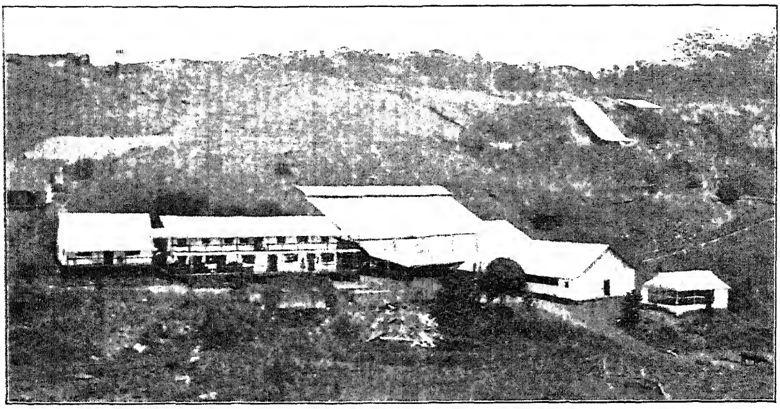


FIG. 5.—BONANZA MINE, PIS PIS DISTRICT.

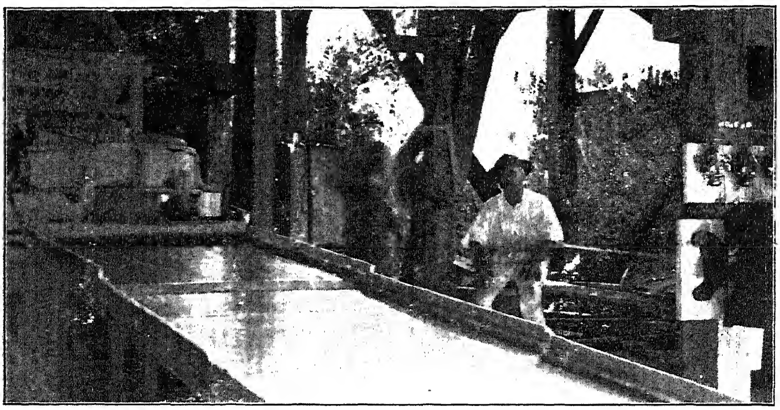


FIG. 6.—HUNTINGTON MILL AND PLATES, MARS MINE, PIS PIS DISTRICT.

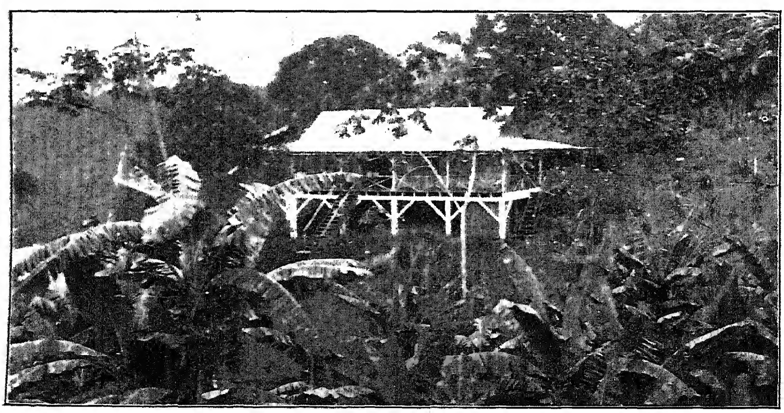


FIG. 7.—SUPERINTENDENT'S HOUSE, SIEMPRE VIVA MINE, PIS PIS DISTRICT.

Difficult though the river-transportation of stores may be, the main trouble and expense begin when the material must be carried overland, from the warehouse to the mine. The freight is transported on the backs of mules, donkeys, or oxen, or else dragged along the ground on wooden sleds. In rare cases, where the conditions are favorable, as at the Santa Rita mine, two-wheeled wagons and frames are used.

Traveling on land in Nicaragua is bad enough when there is no rain; but in the wet season it is an indescribable nightmare. The forests are impenetrable. Instead of walking-sticks the natives carry long knives, or swords, called *machetes*, with which to cut their way through the woods. A path is made through the thick bushes and trees; the heavy-laden oxen follow this path; and soon the hoofs of the animals wear holes so deep in the soil that when you ride along the "highway" on a mule, the poor beast drags his feet from hole to hole, and sinks in the mud up to his belly at every step he takes. If you want to strike a slough of despond, travel on one of these Nicaraguan "highways" in the wet season. You will never forget the experience.

To get freight over such roads is of course very expensive. At first sight it seems impossible; but men have gained experience down there and are expert at the game. Blocks and tackle with heavy ropes are used, together with oxen or mules, in dragging the freight over the mountains. Although mules are more expensive than oxen, I found that it paid to use them. They are much quicker and their backs do not get cut to pieces as easily as the backs of oxen. The ox is a good beast to drag a load, but was not designed as a pack-animal.

The handling of the mules and oxen to the best advantage requires skill and experience. It is most important to have the right kind of pack-saddle (called *aparejo* by the Nicaraguans) and to protect the backs of the animals with the right kind of saddle-cloths. For a long journey a load of 200 lb. is generally put on an animal's back, although for short distances they can carry twice this weight. Care is necessary that the animals are not overworked. When their backs get sore and they are "stove up," it takes a long time to cure them, especially when the *guzanos* (worms) get into the wounds of the animals. It is necessary always to have extra efficient mules and oxen graz-

ing in the pasture, to fill up the gaps made by the incapacitated animals.

While handling freight, the custom is to allow the mules and oxen to scratch around in the bush for food. The indigenous grasses, which are perennially green, are bountiful, but not very nutritious. Wherever possible it pays to feed the pack-animals with corn. The cost of importing it being prohibitive, on many of the properties corn is grown to provide food for the pack-animals.

The oxen and mules raised in Nicaragua are surprisingly sturdy. It is a great cattle-country. Oxen in eastern Nicaragua cost from \$30 to \$40 (U. S. currency) each, while mules range in price from \$75 to \$125. They are brought from the interior provinces of the Republic, for eastern Nicaragua is an uninhabited wilderness, save for the small settlements which have sprung up near the mines. On every property a tract of land is cleared in the forest, to afford grazing for the cattle. No attempt is made to plant grasses in the clearing, and the indigenous varieties spring up in a few weeks.

In the past, little attention has been given to the streams and paths through the forests. The expenditure of a few hundred dollars in blasting out dangerous rocks would have prevented accidents, and if the trails had been better kept up, fewer pack-animals would have been sacrificed. It pays in the long run to make a clearing from 50 to 60 ft. wide through the bush, when the paths are to be used for months. In this way the sun has a chance to get to the earth and dry up the slough of a road. Without such a clearing the path during the rainy season would never have a chance to dry up, for the bush is so thick that in the pristine forest there is a sunless gloom.

In order to do away with packing by mules and oxen, the scheme of a car suspended on a cable, and drawn along by a mule, has often been thought of by engineers, but for long distances this is impracticable. We improved the bad condition of the roads somewhat by placing pieces of timber, from 8 to 10 in. in diameter, across the road and then throwing dirt over these timbers. The expenditure of \$100 per mile in this way makes a big difference in the transportation-account.

In most cases these roads through the forests are made by miners, as the government does practically nothing to open up

or keep in repair these highways. The bulk of the freighting is generally done before the wet season.

VI. Health Conditions.

The popular notion of Nicaragua, and all of Central America for that matter, is that of a death-trap. The awful experiences of the French company in digging the canal in Panama have given the whole country a bad reputation; yet to-day, under modern sanitary science, the Canal Zone has a remarkably low death-rate. With like skill and attention, every other section of Central America can be made as healthy. Yet, on the whole, Nicaragua is not an unhealthy country, even under the present unfavorable conditions. It is nearly free from many of the diseases, such as pneumonia, typhoid fever, etc., which afflict Northern climes, while deaths from consumption are not numerous, and small-pox is infrequent. As for yellow fever, strange as it might seem, no cases are on record as occurring on the east coast. The visitor notices the "yellow-fever mosquitoes"; but as these insects have never become infected from a yellow-fever patient, their bite is not dangerous. Strict quarantine regulations have done much to prevent the appearance of yellow fever in Nicaragua.

It is a mistake to describe Nicaragua as a very hot country. While there is not much difference between summer and winter, the mean temperature is 80° F. But the engineer from the North should realize that he is going to a tropical country, and should adapt his food and clothing to the conditions there.

One of the first things the mining engineer from Alaska will say about the climate is that as a mining-climate it is ideal, for there are no snow-slides or snow-drifts, or frozen water and bursted pipes, to interfere with his work. Mining can be carried on 365 days a year in Nicaragua, so far as the climate is concerned.

Far too much meat, especially imported tinned stuff, is eaten in Nicaragua, and not enough fruit or vegetables. In the past there has been no alternative, when prospectors started into the wilds, but to carry along canned goods. These men do not remain long enough in one spot to start a garden.

But for a man who is to remain for any length of time in the same district, opening up a prospect, the case is different. He

should take seeds with him and make a small garden near his shack. He will be surprised at the rapidity of the growth of his plants, and in a few weeks will be able to vary his meat-diet with fresh vegetables.

Care is required in the selection of the seeds, as the insects destroy many imported plants. I had great success with tomatoes, corn, sweet potatoes, yucca, and beans, as well as with pineapples and bananas. The land gives surprising results with such crops as corn and sugar-cane. Two and a half crops of corn per year can be produced on the same piece of ground, while sugar-cane is planted only once in seven years, the plant growing up like a weed.

There would be less disease were more attention given to diet. In these tropical countries diseases of the liver and stomach are common. In many cases, however, the trouble is not improper diet, but alcoholism. A wise resolution in these Central American countries is to let alcohol alone. Whiskey may be a good drink in Scotland, but it certainly does not suit Nicaragua.

Malaria.

But the bane of the country is malaria. This malady affects people in different ways. Fortunately, the malaria of Nicaragua is not as virulent as in other parts of the world, being milder than the fevers of Central Africa and Eastern Africa, for instance. It does not kill rapidly. If the visiting engineer gets a serious attack of the disease, he has ample time to get out of the country and go to a non-malarial climate, where he can be readily cured. Few maladies are better understood and few yield more promptly to treatment, if the patient escapes from the zone of infection and puts himself under the care of a specialist. Were the visiting engineer more familiar with malaria before landing in Nicaragua he would escape the infection in many cases, but as a rule he knows nothing of the disease when he goes to the tropics. When I went to a malarial country first, I was ignorant of this malady, but as I learned of the fever by having a severe attack, I wish to put down a few facts that I found out about it. To the learned they will seem elementary, but I write for young engineers unacquainted with the subject.

In the first place, malaria is a disease that accumulates in the blood. Unlike typhoid or cholera, one or two injections of the poison into the blood of a healthy, vigorous man will not bring on an attack. It is only in recent years that science has determined the cause of malaria. Formerly it was believed to come from a miasma of the swamps, which rose into the atmosphere with the setting sun. Now it is known that malaria is caused by the injection of poison into the blood by the bite of a mosquito (genus *anopheles*). A man could live in the most fever-cursed swamp in the world and never suffer from malaria if he never got a bite from an *anopheles*. One soon learns to distinguish the *anopheles* mosquito from the other hordes seen in the rivers and swamps. The vast majority of the mosquitoes are of course innocuous; were they all *anopheles*, the traveler would soon die of malarial poison.

The development of the disease is perfectly understood. The microscope is an inestimable help in the study of malaria; and if an engineer would devote a week or so to the examination of slides showing the development of the parasite in the blood, this knowledge would be of great use to him in the tropics, where doctors are scarce, and where he will be responsible not only for his own health but for that of men under him. Under the microscope, one appreciates the deadly effect of quinine on the parasite, and by the use of the microscope one can tell the stage of development of the malarial parasites in the blood, and administer quinine accordingly. Concerning the prophylactic use of quinine there has been a diversity of opinion; but the best authorities now agree that the judicious use of quinine is to be recommended, since thereby many an attack of malaria can be staved off. On the other hand, I have known men in the worst parts of malarial Africa to take from 30 to 50 grains of quinine per day, and yet die of the fever. The reason was, that the infection had gone too far, and the disease had too strong a hold before they took the remedy. The prophylactic use of quinine is not an infallible preventive of malaria; but it does mitigate the severity of some attacks and prevent others altogether. My experience in Nicaragua taught me that when a malarial district is encountered, one should begin taking quinine three times a day, before meals. It is better to spread the doses than to take the quinine at one

time. Three grains before each meal I found to be efficacious.

Quinine, of course, comes in many salts, the one most used being the sulphate. Frequently this is put up in hard sugar-coated pills, which may pass through and out of the system without a particle of the quinine getting into the blood. Such pills should, of course, be cracked before they are swallowed, to give them a chance to dissolve in the stomach. Without such precautions a large amount of the quinine taken may be absolutely wasted. The great objection to the sulphate of quinine is its insolubility. Preferable salts to take to the tropics are the bisulphate and the hydrochloride.

To those going to the tropics for the first time, the following suggestions will prove of value:

1. Don't forget the prophylactic use of quinine in malarial districts. A dose of 3 grains of quinine bisulphate before each meal should be taken regularly. Constipation should be fought, preferably with a fruit diet, but if this is impossible then purgatives, even as strong as calomel, must be used

2. If the fever or chill strikes you, go to bed. Profuse perspiration should be induced by drinking hot lemonade. If the sweating is not copious enough, then some antipyretic, such as acetanilid, antifebrin, or phenacetine, should be given. Ten grains of one of these salts is a dose.

3. If the patient is attacked by violent vomiting, which frequently happens, the best remedy is 30 grains of ipecacuanha, with plenty of tepid water. When the stomach is empty, a dose of 5 grains of cerium oxalate and 5 grains of sodium bicarbonate will bring relief.

4. Treatment with quinine should now begin. From 15 to 20 grains of quinine are given, the dose being administered three times a day. Sufficient quinine should be taken to induce cinchonism. Quinine given in excess of that required to induce moderate dizziness is not necessary.

5. The use of calomel in Northern countries might be open to criticism, but in the tropics, where the liver becomes so torpid, there seems to be nothing to take its place. It is important in the treatment of malaria; 10 grains of calomel mixed with an equal weight of sodium bicarbonate is an average dose. It is best to divide the dose into four parts, and take a quarter of it every 20 min. This prevents the griping pains attending the use of calomel.

6. About six hours after the dose of calomel is taken, an ounce of magnesium citrate dissolved in water should be given; and two hours later a large dose of quinine (say 20 grains) should be administered.

7. The patient should remain in bed for at least a week. Quinine is given two or three times daily to the amount of 30 or 35 grains per day, until 25 days have passed without a return of the fever.

8. At times arsenic, in the form of "Fowler's solution," is preferable to quinine. Five drops in water, three times a day before meals, is efficacious. This tonic should not be taken for more than ten days.

9. When anæmic, iron is necessary. Five-grain pills, containing 2.5 grains each

of iron carbonate and sodium bicarbonate, should be used ; two of these pills to be taken after each meal.

There are, of course, many helpful books on this subject. That of Dr. Scheube, *Diseases of Warm Countries*, published by P. Blakiston & Sons, of Philadelphia, will be found suggestive.

Snakes.

A brief reference to snakes and the cure of their bites is important enough to insert here. In some districts of Nicaragua there are few snakes, while in others there are many. In cold climates, large numbers of poisonous snakes congregate in some cavity in the rocks, during the cold season. In tropical climates, like Nicaragua, this hibernation has an equivalent in æstivation, by which term a scientist has designated a similarly lethargic state during the hottest and driest portion of the summer.

Numerous snakes are found in Nicaragua, including the python, the black snake, the corali, the taboba, and the rattlesnake. In traveling around the country, the engineer should have in his medicine-chest remedies for snake-bite.

In case of a bite from a poisonous snake, one authority[2] advises the following procedure:

> "The first thing to do is to tie a ligature or two between the wound and the heart, next cutting deeply into the punctures so as to make the blood flow freely. Sucking out the blood from the wound is a procedure perfectly harmless, unless the person doing it has an open wound in his mouth. The ligature should be loosened now and then. Small doses of alcoholic stimulant should be given frequently. A tincture of iodine (made in the proportion of one large tablespoon of water to 5 drops of pure iodine) is given to the patient, six drops every half hour.
>
> "For the lesions two or three drops of an aqueous solution (1 to 100) of chromic acid should be injected with a syringe exactly into the puncture of each fang. It is necessary to let the liquid penetrate into the tissues to the same depth as the venom.
>
> "If at the time of treatment the swelling has already obtained a certain size, it may be necessary to make injections into various parts of the tumor. After the injections the part is pressed gently with the hand so as to distribute the injected fluid in all directions and facilitate its mixture with the venom. A small piece of lint soaked in chromic acid solution is applied to the wound."

The Nicaraguans have many remedies for snake-bites, but superstition rather than fact is the basis of their cures. The brain of the snake, dried in the sun, powdered, and then

[2] L. Styneger, *Poisonous Snakes of America.*

swallowed hurriedly, is supposed to work wonders. Some of the indigenous herbs are also used.

VII. Prospecting.

Like most tropical countries, Nicaragua is an exceedingly difficult one to prospect. By far the best prospectors are the natives, and the foreigner can do no better than to attach himself to one or two of these gold-seekers, and open up the places which the natives point out as containing gold. No one knows the forests and streams as do the Indians, whose keen eyes are ever on the lookout for gold-bearing rock. Most of the mines opened by Europeans have been found by following the clues which the natives have given. So thick is the bush, and so deep is the mixture of soil and wash covering the formations, that the prospector accustomed to open countries like our West, or South Africa, feels lost for a while. However, he soon learns to take advantage of the exposures made by the numerous streams, and looks for gold first along the banks and river-beds. An occasional piece of flat rock containing gold, at the bottom of a hill, is a sign for him to inspect the hill carefully. This he does, cutting his way through the bush, and panning as he ascends the mountain. In the majority of cases he does not find gold in payable quantities. Occasionally, however, his search is rewarded by the discovery of a vein from 30 to 60 ft. wide, like those of the Bonanza, Lone Star, and Siempre Viva mines.

The use of the pan and the horn spoon is of course common in Nicaragua. Far too much reliance is put on them, and it is forgotten that at the best the pan is but a rough guide, and no estimate should be made of a mine or a prospect until the samples are assayed.

Several disappointments are due to this neglect. In one case known to me, the mine was tested almost entirely with a pan. The man who did the panning was accustomed to gold far coarser in grain and finer in grade than the metal he found in Nicaragua, where he forgot to allow for the extra amount of silver found in the ore. No wonder, therefore, that when the little mill commenced work the amount of fine gold caught was far below expectations. On account of this mistake in calculation, the enterprise was a failure.

But for the great concessions granted in the past, there would have been far more prospecting in Nicaragua. Men who have gone into the wilderness have been successful, as such mines as the Bonanza, Lone Star, Concordia, etc., testify. The danger in Nicaragua is that the prospector may be too optimistic, and think he has a mine because of the rich pannings from the *manto*, or part of the vein which has been removed by erosion.

No attempt has been made to locate mines which would require shafts; and for this reason the prospectors have confined themselves to the hilly country, where adits can be run and a mine can be opened up cheaply. The problems of shaft-sinking and pumping have so far seemed to frighten the prospectors. The deepest vertical shaft in eastern Nicaragua is at El Paraiso mine, in the Oconguas district, where a depth of 130 ft. has been reached.

Figs. 4 and 5 are views showing the Lone Star and the Bonanza mines. The type of mill and plates in use at the Mars mine is shown in Fig. 6. Fig. 7, a view of the house of the superintendent at the Siempre Viva mine, also shows the character of the vegetation of the region.

VIII. Geology and Vein-Phenomena.

The geology of Nicaragua has not been thoroughly studied. The visiting engineer soon notices that the country is made up almost entirely of igneous rocks. He sees few sedimentary strata. Belts of limestone are occasionally met; but they are not extensive, and have been tilted by the igneous rocks. Signs of vulcanism are more striking in western Nicaragua and around the lake than in the eastern section. The several volcanoes, some of which are still active, are in the western part of the country. It was the fear of volcanic and earthquake disturbances which had so much weight in deciding the engineers against the Nicaraguan route for the Canal. The mining engineer sometimes wonders whether earthquake "trembles" would not undo all his work in the mines.

On the high summits of the mountains are seen overflows, mostly basalt, tufa, etc. The evidences of vulcanism are strong in western Nicaragua. Sedimentaries are unknown in this

section. In the low ground, rhyolite sheets are noticed; also andesite, and sometimes diabase.

In eastern Nicaragua, the center of vulcanism being so far removed, the formations are less disturbed. Although the rocks met are mostly porphyries, areas of limestone are also seen. In the Pis Pis district, the best-known section in the northeastern part of Nicaragua, the country-rock generally is porphyry. One is surprised to see how infrequent are large faults. Large veins, 40 to 60 ft. wide, run for several miles in the Pis Pis district with scarcely a throw. Dikes also are rare. In these large veins frequent "horses" of porphyry are found. As a rule, it is necessary to mine them along with the ore. At times the "horse" becomes sufficiently thick to permit the two sections of the vein to be mined separately. The large amount of porphyry that goes to the mill causes the slime to be troublesome.

The rainfall is so heavy that the denudation of the outcrops of the veins has been great. This denuded auriferous material has in some instances covered the hill-sides, and formed small placers in the valley and along the stream. In this way has been formed the *manto* on the hill-sides. This *manto* varies in width from a few inches to several feet. The gold-values are very erratic, and while on some mines it has paid to put the *manto* through the mill, as a rule it could be worked to more advantage by hydraulicking when water is plentiful.

In panning this *manto* I have found a curious occurrence of gold on one property, namely, in the form of fine wire, from $\frac{1}{8}$ to $\frac{3}{8}$ in. in length. I found that this "wire-gold" was very base, the alloy being copper.

The Bonanza mine, Fig. 8, furnishes an interesting example of *manto*, showing that under favorable circumstances it may be valuable. At one time the outline of the surface was probably as shown at *A B E F D*. Denudation cut out *F C D*, and in time *G E F C* fell towards *C D*, forming a thick *manto*, which it has paid to work.

On account of the intense weathering and rainfall, the distance to permanent water-level in Nicaragua is far less than in a semi-arid country like Arizona or the Transvaal. The vital question, what will happen to the ore-bodies at water-level and below? is one that the engineer soon asks himself. The possi-

bility of an impoverishment of gold, and an increase in the base metals, so as to make the gold-deposit unworkable by milling methods, is one that he keeps in mind.

The indications in many of the mines point to the appearance of a large increase of sulphides at water-level and below. That the veins will hold to considerable depth is almost certain. Ore-bodies of such length and breadth will not die out suddenly. But the question is, will not the appearance of sulphides of copper, lead, and zinc below the water-level change the deposits from milling-cyaniding mines to "smelting propositions"? It will probably pay some day in a developed district to have a central smelter, although it is not encouraging in a new mining-country to contemplate changing the process of ore-treatment so soon after water-level is reached. It is

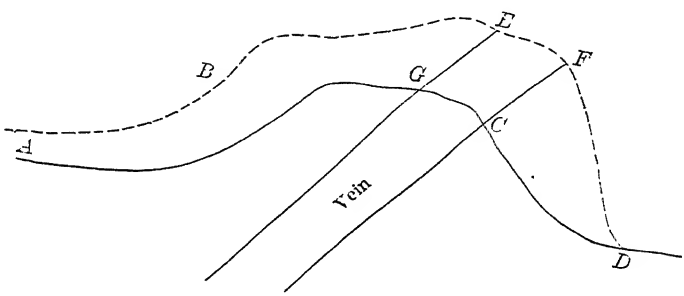


FIG. 8.—VEIN AT BONANZA MINE, PIS PIS DISTRICT.

probable that a combination process of crushing and extracting the concentrates for shipping, and then cyaniding the tailings, will prove satisfactory.

Most of the mines have so much ground on the strike of the vein that the problem, what will happen below water-level, has never bothered them. It is asserted that there is enough ore above water-level to last for many years. Take the case of a company owning a mile along the outcrop of a vein 40 ft. wide. If water-level is assumed, say, at 360 ft., then on this vein, leaving out the "spurs," the owner will have between six and seven million tons in the oxidized zone. Of this total, say, five million tons will go to the mill, which is crushing, say, 30,000 tons per year. Then there is enough ore to keep the mill running for 166 years. No wonder the owner of the mine, with a

small mill, has refused to worry about the ore below water-level! But for a company putting up a large plant, the position is different. On account of the probabilities of the change in depth, it is important to acquire as much of the outcrop as possible, so that a large tonnage may be developed in the oxidized zone. Mines like the Bonanza, in the Pis Pis district, have sufficient ore on the large veins above water-level to keep a good-sized mill running for years.

Vein-Phenomena.

There are veins and deposits of many types in Nicaragua. It is this variety which makes the country so interesting to the engineer. In the Pis Pis district the veins are generally wide replacement- and fissure-veins in porphyry, assaying from \$4.50 to \$7 per ton. In the Bana Cruz district, about 16 miles

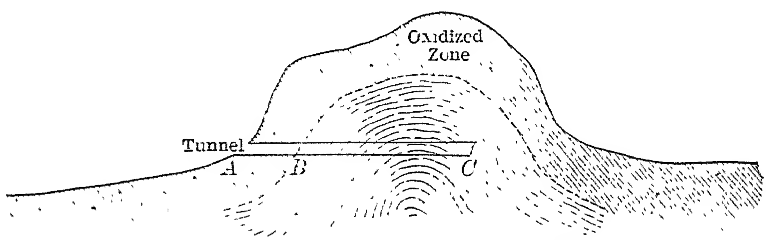


FIG. 9.—ROUGH SECTION OF SANTA RITA MINE.

south, the veins are not so wide, but of higher grade. Here the "country" is mostly porphyry also.

A deposit of an entirely different type is the Santa Rita mine, Fig. 9, in the low swampy land on the Bambana river. Here a hill of porphyry rises abruptly from the flat country. Through the hill there are numerous little veinlets, running through the porphyry in every direction. These veinlets are very erratic, and the only way to work the formation is to mine the hill by the "glory-hole" method. This of course means a low value per ton, but also low working-costs. The ore is soft and porous, some of it floating in water, and is ideal for a Huntington mill. A daily tonnage of about 75 tons is run through a 5-ft. mill.

In going into the hill, it was found that the character of the ore changed rapidly, copper sulphide appearing, and the gold-

content of the ore falling rapidly. Only the outer crust of the hill could be worked for its gold-ore.

The tunnel from *B* to *C*, Fig. 9, opened up some good copper-ore, carrying from 8 to 10 per cent. of copper. On account of the cost of transportation this copper-ore cannot be worked. Whether it will hold out in depth remains to be proved. The chances are that it will be largely replaced by iron sulphide.

A formation that is liable to fool the prospector, I found in the Oconguas district. It is illustrated in Fig. 10. In prospecting along the stream *AB* I got some rich pannings. The gold was rather angular and I decided that it had not traveled far. I proceeded up the hill to *C*, panning the *manto* and getting "colors" all the way up. At *C*, I found a veinlet about 15 in. wide and very rich in gold. Trenching along its outcrop showed that it extended about 100 ft. and then pinched out

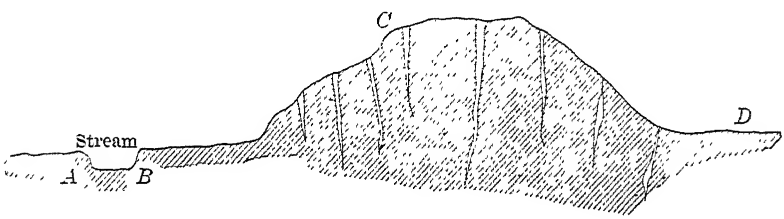


Fig. 10.—Formation in Oconguas District.

altogether. A winze sunk on the veinlet showed that it "petered out" entirely at 25 or 30 ft. Further work showed that small veinlets of this kind ran through the porphyry, and that the gold in the *manto* and the stream came from these *elos*, as the Nicaraguans call them, and not from a large, well-defined vein, such as I had hoped to find. Although I got remarkably rich pannings from the hill, the discovery was of no economic importance.

One of the most interesting occurrences of gold which I saw is illustrated in Fig. 11. In the Oconguas district there is much limestone, containing numerous gold-silver veins, usually narrow (from 12 in. down to a knife-blade). The El Paraiso vein is the one which has been most worked in the district. On the surface thousands of dollars were taken out of the rich *manto*. The vein varied in thickness from a few inches to $\frac{1}{8}$ in. There are pockets of ore so rich that it is brought up in a bucket

and ground in an old *molino*—a system used centuries ago. The owner does well even with this crude method. Apparently there is a well-defined fissure, *AB*, in the limestone, along which the gold has been deposited. The richest ore comes from the small bulges in the limestone, as at 1, 2, 3, 4, 5. Here the gold occurs associated with calcite crystals, frequently as a thin film on the face of the crystal. The proportion of silver in the ore is high. The indications at the bottom are that this vein continues in depth.

About 12 miles from El Paraiso is La Luna mine, where entirely different conditions obtain. The country-rock is porphyry. The foot-wall is well defined, being as smooth as glass,

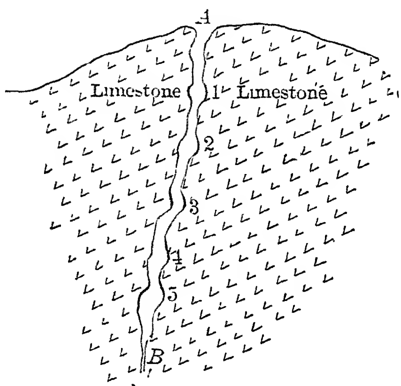


FIG. 11.—SECTION OF EL PARAISO VEIN, OCONGUAS DISTRICT.

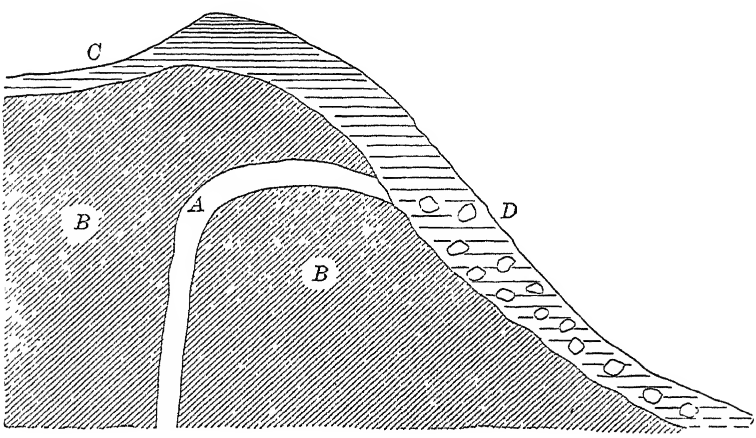
and was undoubtedly the line of ascension for the auriferous solutions. The replacement of the country-rock is not well defined, and in consequence the hanging-wall is not sharply drawn between ore and country-rock. The workable width of the vein is 33 ft. Small "horses" of porphyry, a foot or two in width, occur. The vein-material itself is quartz, and the values are erratic; the average value in gold being $9.50 per ton.

Fig. 12, showing a vein in the Chontales district, is taken from *The Naturalist in Nicaragua*, and represents a face in the mine worked in 1869–71.

To show the preponderance of gold in Nicaragua, it is interesting to note that out of 500 mines registered in the Bureau

of Statistics of the Republic, 494 are producers of gold. Among the properties in the hands of British capitalists are the Santa Francisca, San Lucas, Amoya, San Cristobal, Quilala, and San Juan Tetelponeca (on the west coast). The Leonesa mine, purchased some time ago by H. C. Hoover for London capitalists, promises to become a fine property.

On the west coast some deposits of silver-lead have been found. It is said that coal has been discovered in the Sequia district not far from Roma, but little is known of it. No proved deposits of copper are known. It seems almost certain that gold will be the mainstay of mining in Nicaragua in years to come. In normal times the value of the export of gold is about $1,000,000 per annum.



A. Vein; *B.* Dolerite; *C, D.* Manto with auriferous quartz boulders.

FIG. 12.—SECTION OF VEIN ACROSS SAN ANTONIO VEIN, CHONTALES DISTRICT.

IX. WORKING-COSTS.

In the Pis Pis district and in other districts with similar veins, the total working-costs, including development, milling, cyaniding, etc., are from $2 to $3 per ton. The cost for mining varies, according to the facilities of working, from $0.50 to $1 per ton. Milling and cyaniding amounts to about $1 per ton, maintenance to $0.20, and general expenses to $0.48.

At the Siempre Viva mine, the total expenses are divided as follows: wages to all employees, 48 per cent.; boarding em-

ployees, 23 per cent.: supplies, 23 per cent.: and bullion expense, 6 per cent.

At the Lone Star mine, where the conditions for handling ore are more favorable than at Siempre Viva, the costs vary from $1.75 to $2.50 per ton. The 30-stamp mill crushes about 90 tons daily. The detailed cyaniding-costs on the Lone Star are as follows per ton of ore:

Potassium cyanide.	$0.228
Caustic soda,	0.188
Zinc,	0.064
Labor,	0.092
Superintendence.	0.178
Total,	$0.750
Smelting, etc.,	0.035
Government fees.	0.101
Total,	$0.886

These low costs are attained with small mills, none of them exceeding 100 tons daily capacity. With improved conditions, such as larger mills, railroads, air-drills, etc., costs in the Pis Pis district could be cut nearly in half, making it one of the leaders of the mining world in this respect. Even now, these low costs, in an undeveloped country like Nicaragua, are a great surprise to the visiting engineer.

Transportation is one of the principal items of expense. For this reason far more attention should be given to agriculture, in order to reduce to a minimum the transportation of food-stuffs. Many of the mines have plantations on which are grown bananas, plaintains, cassava, corn, etc. No scientific agriculture exists. No plows are used to break up the soil in planting corn. In the wilderness, after the trees are felled, a man takes a sharp stick, pokes small holes in the ground, and puts in a few grains of corn. After that the crop must shift for itself, no attempt being made to hoe away the grass or weeds that spring up with surprising speed. Not only is Nicaragua a mineral country, but it is also a remarkable agricultural country. Yet many difficulties are encountered in growing food-stuffs for the mines. First there are numerous rats that eat up the seed. This pest can be exterminated by placing in the bushes cracked corn soaked in strychnine. The rats die

so rapidly that the survivors take fright and depart. Then the ants threaten to clear everything before them. A constant fight must be kept up against these little creatures, but patience and perseverance will overcome them. If carbolic acid mixed with water is poured down the burrows, the ants will soon shift their homes. Corrosive sublimate sprinkled across the path of a column of ants has a remarkable exterminating effect. Spreading coal-tar across the paths leading to the gardens is also efficacious as a protection. Only one who has tried to grow plants in Nicaragua can appreciate the power of the ants to thwart all his efforts. It is well to stick to those crops which experience has proved can be readily raised in spite of the ants. Among the best are plantains. They are not as quick-growing as bananas, but are much more highly appreciated by the Nicaraguans, who consider them excellent food.

The cost of meat can be greatly reduced by installing an ice-plant at the mines. This has been done at the Bonanza mine, where it is consequently possible to kill a steer and keep the meat for days, and not depend upon the man who has a "concession" to supply the district with beef. By buying cattle at a low figure in the interior, driving them to the mines, fattening and killing them, and keeping the meat in a refrigerator, the price of meat is reduced to one-third of what the concessionaire demands for his beef. Also, the freight on tinned meat is saved, and the workmen are better satisfied.

As yet no air-drills have been used on a working-scale. At the Lone Star, a couple of drills were run for a short time, and they were tried at the Topaz also; but the full advantages of drills have not been appreciated yet. With hydro-electric power, the use of air-drills in stoping and development-work would reduce the costs.

An explanation for the low working-costs is the fact that the ore comes through adits, and there is no charge for pumping or hoisting. Moreover, the veins are wide and rather soft, so that mining costs little. Finally, the ore is crushed cheaply and the consumption of cyanide is small.

Water-power is used at all the mines which work cheaply. Without the abundance of water-power with which Nicaragua is blessed, only a few mines could work. For a short time, fuel is not expensive; but it does not take long to clear the wood

away from the mill-site, and when wood must be brought from a distance through the Nicaraguan forest the cost is almost prohibitive.

On some mines, as at the Lone Star, the water is used on a Pelton wheel, which drives all the machinery direct. At other mines, like the Siempre Viva, there are fine water-falls within a few miles, and the power-scheme is hydro-electric. Pelton wheels under a head of a few hundred feet are used. I know of no turbines.

The one objection to water-power is the fact that the smaller streams are liable to run low in the dry season, in March or April. Unless an auxiliary steam-plant is provided, the small mill will be forced to shut down for perhaps six weeks. To build a storage-dam for water, for use during the dry season, is in most cases impracticable, on account of the precipitous nature of the country, and the cost of a dam which would give enough water-power during the brief dry season is prohibitive. In order to run the mill all the year, it is necessary in many cases to have an auxiliary steam-plant.

When the electrical transmission-line is of any length, the trees should be cleared away, and the poles put down substantially. On account of the destruction of most of the woods by the ants and other insects, it is cheaper, for a plant that is expected to run a long time, to put in steel poles. The trees in the wilderness do not send down heavy tap roots, as in the northern countries, and are readily blown over by ordinary storms. It is necessary, therefore, to cut down the trees in the neighborhood of the power-line, to prevent them from falling on the wires.

Occasionally the ubiquitous monkeys try to flirt with the high-voltage wires, and sometimes, when a short circuit is made and all the fuses are blown out, investigation shows a frizzled monkey between the wires.

X. Timber.

Until he learns the forest, the new-comer makes the mistake of thinking that the percentage of wood suitable for fuel and timber is high. He sees trees everywhere so thick that he can scarcely crowd through them. By and by experience teaches him to divide the timber in the Nicaraguan forests as follows:

about 8 per cent. of the trees are the finest timber in the world; 7 per cent. are good, if protected from the weather; 10 per cent. can be used, but are not good timber; 35 per cent. are available for fuel; and 40 per cent. are mere rubbish, good for nothing.

The mahogany and hard woods for which Nicaragua is celebrated occur in scattered patches. In the virgin forests, there is plenty of this fine timber for mine-use, and for the construction of mills and buildings. In view of the price paid for mahogany and hard woods in other parts of the world, it seems extravagant to put this timber into mills, buildings, and even as fuel under boilers.

There is one wood—that of the mangley-tree—so full of tannic acid that when some of the bark and branches are put into a boiler, the formation of scale is prevented. I have seen boilers run for months and kept clean, not with any boiler-compound, but with pieces of mangley bark thrown into the boiler-water.

The outlook for the discovery of coal in the country is poor. Should there be a great industrial development in Nicaragua, those mines which have not water-power must import coal in the course of time, probably from Alabama.

Rubber-Trees.

It often occurred to me that rubber-growing and mining could be carried on simultaneously in Central America. It must always be remembered, however, that rubber is a wild plant. The attempts to grow it under cultivation in cleared land have not been overly successful in Nicaragua. The rubber-tree must have plenty of shade. It should, therefore, be planted in the forest and not in cleared land. Of course, this would require extra labor and attention, and there would be no return for a number of years. But we all know how rubber has advanced in price. It grows well in Nicaragua and is one of the valuable exports of the country. It requires about seven or eight years for the trees to commence producing rubber, and to return a large interest on the investment. Nicaraguan rubber stands high in the markets of the world, and rubber-growing, properly managed, is a safe investment. I believe the combination of mining and rubber-growing would prove a profitable alliance.

XI. Labor.

While labor is not plentiful in eastern Nicaragua, there has been, so far, no trouble in supplying the demand. With an expansion of industry in Nicaragua, labor will probably become scarce for a while, but it must be remembered that it is not far to regions like the West Indies, etc., where labor is abundant.

Labor in Nicaragua may be divided into four classes:

1. The so-called "Europeans" (mainly Americans and British), who do the skilled labor, as managers, cyaniders, mill-men, etc. One seldom sees white men at work underground. They prefer prospecting to working in the mines.

2. The so-called "Spaniards," *i. e.*, the Nicaraguans, who do most of the mining. These men do better work than the Mexicans, and have not developed the "*mañana*" fever as badly as the Spaniards further north. Interruptions in the work, due to holidays and Saints' days, are not as frequent as in Mexico. Working on contract I have seen these Nicaraguans do as good work as any miners in the world. The contract-system, especially in development-work, is by far the best to use with them. Working by the day the miners receive from $1 to $1.25 U. S. currency per day, while on contract they earn from $1.60 to $4 per shift. The price per foot given on contract varies, for "picking-ground," from $0.33 to $0.65; for hard rock, from $1.60 to $2.60.

By working three shifts the drifts can easily be advanced 45 ft. per month, when the men work on contract.

The "Spaniards" make good mill-hands, although it is wise to have a white man as superior in the mill. Almost all the Nicaraguans live in the interior of the country. Honduras and Costa Rica also supply some of the miners. In the future more well-built villages should be started in the mining-districts, to induce the miners from the interior to move and live near the mines with their families.

3. Next to the Nicaraguans in importance are the Jamaicans. At Bluefields and along the coast there are large numbers of Jamaican negroes, but in the cities of the interior, Granada, Managua, etc., one seldom sees them. As a rule, they are not liked as laborers, being generally lazy, incompetent, and inso-

lent, although there are exceptions, of course. Few, if any, Jamaicans are found mining, their preference being for carpentering, operating machinery, etc.

4. The fourth class of laborers comprises the Indians of the Mosquitoe and Sumu tribes. Outside of their work as paddlers on the rivers, they do not shine as laborers. Generally they are employed as muckers, wood-choppers, etc., and are paid from $0.40 to $0.80 per day.

There are different preferences for laborers. Some like to have all Nicaraguans; a few operators employ all Indians; but the best scheme is to mix the three, and work Indians, Spaniards, and Jamaicans.

The feeding of the laborers in a country like Nicaragua is expensive. Not only is the freight-rate high (from 2 to 6.5 cents per pound, from Bluefields to the mine), but the terrific duties imposed on clothing and food-stuffs make the commodities most costly.

The chief foods are beans and rice, both of which can be grown in Nicaragua, but are generally imported from the United States, the beans from California and the rice from Louisiana. At the Lone Star mine in the Pis Pis district, where freight from the coast is from 4 to 6 cents per pound, the following rations are allowed per man per week:

	U. S. Currency
8 pounds flour at 11.25 cents per pound, equals,	$0.90
4 pounds beans at 12 cents per pound, equals,	0.48
3 pounds rice at 14 cents per pound, equals,	0.42
1.5 pounds lard at 22 cents per pound, equals,	0.33
1 pound salt at 10 cents per pound, equals,	0.10
1 pound sugar at 16 cents per pound, equals,	0.16
1 pound coffee at 22 cents per pound, equals,	0.22
4 pounds beef at 14 cents per pound, equals,	0.56
Total,	$3.17

On an average it costs 50 cents U. S. currency per man per day to feed the laborers. On comparing this figure with the 6 cents per day for feeding Kaffirs in South Africa, and the 12 cents for feeding the Chinese coolies on the Rand, one sees how large is this item in Nicaragua.

XII. Metallurgy.

On the whole, the metallurgy of the gold-ores so far worked is not difficult. Now and then complicated ores are encountered; and unless they can be treated by some wet method at the mine, they are of no present economic use. With such a high rate for transportation, ores must be rich to stand shipping to a smelter in the United States or Europe.

The ore as it comes from the mine is put through a Blake, Gates, Dodge, or similar crusher and fed to the stamp- or Huntington mill. In general, fine grinding is not necessary, for the ore is rather porous and gives a good extraction with cyanide, even when a coarse screen is used. Using a 20-mesh screen on the Pis Pis ore, I have obtained an extraction of 86 per cent. on $2.10 ore.

I believe the Huntington mill is generally better suited to the ores of Nicaragua than the gravity-stamp. At some mines the gravity-stamp is almost useless, because the ore is so "muddy" and sticky that the shoe of the stamp becomes covered with the material, and little is splashed through the screen. To make the stamp heavier only aggravates the trouble.

Even in harder ore there are streaks of soft material which retard the speed of crushing with stamps. For these ores, made up of hard and soft constituents, the best method of crushing would be, first, through a Dodge crusher, then through rolls, and then through the Huntington mills. In other words, crushing by stages would be the best scheme for the average Nicaraguan ores.

As a rule, the ores in the Pis Pis district are readily amenable to cyanide treatment. Now and then a complex ore is met, that requires some study. I heard of one case which interested me. The owners of a mine with a narrow, rich vein were about to despair of the enterprise because they could not extract the gold by stamp-milling and amalgamation. Experiments made in a strong cyanide solution on ore that had been crushed between rolls showed that by this treatment the ore would yield a good profit. By sorting, the grade could be raised to $47 in gold and 11.6 oz. silver per ton. The vein is in an andesitic country and carries pyromorphite, chalcopyrite,

marcasite, and sphalerite. By grinding through a 60-mesh screen and consuming 6.5 lb. potassium cyanide per ton treated, it was possible, after nine hours of agitation, to get an extraction of 94 per cent. of the gold and 70 per cent. of the silver.

One reason for the failure of many gold-mines in Nicaragua has been the low extraction of gold. On some properties 35 per cent. is the total obtained by simple amalgamation. On mines in the Pis Pis district, cyanide-plants have been put in, and the gold from the sands is recovered; but with the exception of the Bonanza mine, no attempt is made to treat the slimes. Since from 45 to 55 per cent. of the pulp from the mill runs away as slime, it is evident that there must be great losses where slime-plants are not installed.

On the Bonanza, about the worst possible slime-process was put in, namely, the decantation-process, and it has proved an awful mess. This is one of the few places in the world outside of the Rand where the clumsy, expensive decantation slime-process has been installed. It just about pays expenses. The extraction at the Bonanza is claimed to be from 83 to 86 per cent.; on mines without slime-plants the extraction ranges from 55 to 70 per cent. In these days of advanced gold-metallurgy, these extractions are far too low. Fancy running mines in Africa, and obtaining no more of the gold than they extract in Nicaragua!

The sands, as a rule, give no trouble in the cyanide-treatment, provided great care is taken to keep slimes out of the leaching-tank. On one or two occasions, the cyanide-process has been declared a failure in Nicaragua because the man using it ran all the mill-pulp into one tank and then wondered why he could not get the solutions to leach through in a month.

The slime-problem is far more difficult. It will never be solved by the South African decantation-process. Laboratory-tests on typical tailings of the Pis Pis district show an extraction exceeding 90 per cent. on from 20- to 30-mesh material in 14 hr. Fine grinding is not necessary.

The serious trouble begins with the settling of the slimes and the clarifying of solutions. In company with T. W. Bouchelle, of the Lone Star mine, I carried out a number of experiments on the slime. We found that as they come from the mill, before any alkali (caustic soda or lime) or cyanide is

added, the slimes settle rapidly. As soon as the chemicals are put in, however, conditions are changed.

We allowed slime, neutralized with soda or lime and treated with cyanide, to stand for days, without getting a settlement. We filtered the solution through three thicknesses of filter-paper, and still the liquid came through opaque. In precipitating the gold from the solution with zinc, this finely-suspended material covered the zinc.

The trouble is due to the reactions between the high percentage of alumina in the ore and the chemicals added, a large quantity of aluminum hydroxide being formed. We came to the conclusion that the slime-problem can be readily solved in Nicaragua by using one of the standard filter-press methods, and taking especial pains to clarify the gold-bearing solution from the slime by sand-filtering, etc. Should it be impossible to clarify the solution by this method, a perfect precipitation of the gold from the solution can be obtained by the electrolytic method. The slimes assay from $2.25 to $3 per ton and can be treated for $0.75 per ton, with an extraction of 90 per cent. It costs about the same to treat the slimes as the sand.

A large company starting operations in Nicaragua must attack the slime-problem, and stop the enormous waste from this product, which for years has run down the rivers to the sea.

Along the coast one notices the millions of tons of coral-reef, which on burning makes excellent lime for use in the cyanide-process. Instead of importing any lime, this coral-reef can be used.

On account of the high freight on lime, caustic soda is used for neutralizing, the salt being spread on the top of the sand-tanks and not added to the sumps. Cone-separators are used to take out the slime from the sand. The sand is automatically distributed in the tanks, which are from 4.5 to 5 ft. in depth and 20 ft. in diameter. The strength of cyanide solution used is about 0.25. The ore being rather porous, a good extraction (from 87 to 89 per cent.) is obtained from the sand even with a 20-mesh screen. The time of treatment is four or five days. At the bottom of the tanks a heavy canvas duck is used, instead of the cocoanut-matting employed in many parts of the world, to filter the solutions. The canvas is kept taut

by means of a rope, hammered down around the circumference of the bottom of the tank. This canvas as a filtering medium is sufficient, together with a small settling-tank, to deliver the cyanide solution to the zinc-boxes in a well-clarified condition. Ordinary precipitation-boxes and zinc-shavings are used.

Occasionally copper and arsenic are found in the ore, as well as zinc and lead sulphides, and then the consumption of cyanide increases, and trouble is likely to occur in the precipitation-boxes. By following the advice given in Park's book on the cyanide-process, namely, to use mercury bichloride in the zinc-boxes, ore carrying from 0.5 to 1 per cent. of copper is readily treated.

When arsenic became troublesome, Mr. Bouchelle informed me that he overcame the difficulty by the use of steam. A pipe is placed in the zinc-box and the solution heated up. The heat causes the arsenic which has deposited on the zinc to crack and roll off, leaving the zinc free for the deposition of the gold.

As a rule, the tanks used in the cyanide-plants are of wood. At the La Luz y Los Angeles mines, however, they have erected fine steel tanks. The tanks are generally imported from the United States in a "knock down" condition and erected at the mines. In most cases it would be cheaper and better to put up a small saw-mill and cut the timber for the tanks.

Without an assay-office the cyanide-process cannot be very well carried on. In this regard the mines in the Pis Pis district are well equipped. All of them have put in the Braun gasoline-furnace, without which assaying would be very difficult in such an out-of-the-way region as Nicaragua. If it were not for these gasoline-furnaces, charcoal made from the native trees would have to be used, as it would be out of the question to import coke.

On account of the humidity of the atmosphere, the assay-balances, surveying instruments, etc., rust rapidly; and for this reason it is necessary to oil and clean them frequently. A mildew collects on all books during the wet season, and to protect his technical books and note-books, the engineer must wrap them in cloth.

XIII. Conclusion.

In this paper I have been able only to touch the subject of mining in Nicaragua. I have tried to bring out the difficulties in the way, as well as to show the advantages of the country. Of course, so far, the industry in Nicaragua is in its infancy, and the chief difficulties of deep mining, such as handling large quantities of water, have not been encountered. Deeper mining will mean higher working-costs; but I believe more efficient methods and larger mills will counterbalance that increase. For many years to come, the total working-costs in Nicaragua should not be more than $2 to $3 per ton.

It exasperates the engineer to travel through Nicaragua and other parts of Central America, and to see such vast potentialities, in agriculture as well as mining, lying dormant. Nicaragua has the makings of a great country. The world is growing too crowded and too civilized to allow such fair prospects to lie idle indefinitely. The completion of the Panama Canal will direct the attention of the world more and more to Central America.

Nicaragua, named after an Indian chief, has an area of 49,200 sq. miles and a population of only 600,000. It is therefore the largest of the Central American republics, and, with the exception of Honduras, the least populous. Financially, the country is not in bad condition. In 1909 the total outstanding foreign public obligations were $3,875,000 and the internal debt $5,127,000. Compared with Honduras and its debt of many millions, Nicaragua is in a happy position financially.

I feel confident that better days are in store for Nicaragua. The Atlantic coast of the Republic is a most promising mining-region, as well as the other provinces; but their exploitation requires plenty of capital, as well as skilled and practical knowledge.

The Condensation of Fume and the Neutralization of Furnace-Gases.

BY F. T. HAVARD.

(Canal Zone Meeting, November, 1910.)

I. Introduction.

The present truce in litigation between Western smelting and ranching interests gives opportunity for a summary of the results achieved by metallurgists in condensing fume and de-acidifying furnace-gases. In the absence of complete records of past experiments, there has been a good deal of unnecessary duplication of attempts and failures. Yet, especially under the recent policy of our large corporations, looking to the economic recovery of the values of by- and waste products, original and, in some cases, effective measures have been developed by American metallurgists.

In the case of Bliss *vs.* the Anaconda Copper Mining Co., the Montana Supreme Court, recognizing the progressive efforts made to render the flue-gases innocuous, and the value to the community of the smelting company's operations, refused to hamper its work, and returned a verdict in favor of the defendant. In the trial, the experimental farms of the company in the so-called "affected region" were proved to be thriving perfectly, showing that immunity was guaranteed to vegetation and stock by the wide distribution of the sulphurous acid gas in the dry air of Montana. Indeed, L. S. Austin has proved that by the time the fume is 4,000 ft. distant from the stack, the percentage of sulphurous acid has been diminished from 2.28 to 0.00045; and Angus Smith has shown that rain must contain 40 parts per million of free acid before vegetation is affected. The arsenic in the stack-gases is but a very small part of the original content of the ore, of which 80 per cent. and over is rejected in the concentrator and furnace waste-products or recovered in the flues.

Nevertheless, neither the peace in Montana nor the armistice

between the Salt Lake smelting companies and the neighboring farmers of Utah should lull us into false security, while we have still before us the troubles of such plants as Tarnowitz, in Germany, and Selby, in California. Few smelting-districts are blessed with the dry air which favors, at the Montana smelteries, the distribution of furnace-fume over a wide area, and the minimum production of hydrous acid flue-gases.

Fortunately for the German metallurgists, Tarnowitz, Clausthal, Freiberg, and Oker are all fiscal works; therefore, the governments of Prussia, Saxony, and Brunswick-Hanover had a true appreciation of the difficulties involved in removing the sulphurous acid, and made regulations permitting a reasonable quantity of this gas to pass out of the stack. In no case, so far as I know, has any smelting-establishment in Germany been closed on account of alleged hurt to forests or pastures. In England, no limit is placed on the content of sulphurous acid, but not more than 1.5 grains of sulphuric acid per cubic foot of gas is permitted to escape from the stack. The U. S. District Court, on the other hand, in settling the Bingham Valley dispute, placed the limit of the sulphur dioxide content of stack-gases at 0.75 per cent. and insisted on the neutralization of the sulphuric acid in flue-gases.

Naturally, the problem before the smelter varies with the kind and quantity of ore treated and the locality of operation. Thus the lead-smelter can direct all his energies towards neutralizing the acid sulphur-gases, since he may rest content with the efficiency of the modern bag-house as a collector of dust, metallic lead, silver, antimony, and arsenic. The iron-smelter, by utilizing the thermal value of the gases which have been freed of dust in separators, spraying-towers, and Theisen centrifugal washers, has met the demands of modern technology. The antimony-smelter has nothing to fear, since of all the metallic fumes, antimony is most easily recovered by condensation with water, applied either in spraying-towers or in centrifugal cleaners. It is the smelter of large tonnages of sulphide and pyritic copper-ores who faces the most difficult problems of condensation and de-acidification.

E. P. Mathewson succinctly reviewed these difficulties in his evidence before the Conference of the Counsel with Judge Hunt in the case of Bliss *vs.* the Anaconda Copper Mining Co.

The possible methods by which arsenic might be extracted from the flue-gases at Anaconda, he classified as follows:

1. Cooling-processes: (*a*) water-spray; (*b*) admission of air; (*c*) radiation; (*d*) freezing.

2. Filtering-processes: (*a*) bag-house; (*b*) friction; (*c*) centrifugal gas-cleaners.

3. The Cottrell method.

In commenting on the spraying-method, Mr. Mathewson emphasized the difficulty of disposing of the acid mud formed, and said that a plant sufficient for the Washoe smeltery would cost $3,000,000. He disposed of the bag-house method by saying that a sufficiently capacious plant would cost $2,750,000; that its operation would cost $1,850 per day, with a recovery of arsenic worth only $204; and that the life of the bags would be short. The installation of a radiation cooling-system would cost, in his estimation, $1,200,000, and would not be efficient. For the freezing-system, the pipes alone would cost $4,000,000, and the cost of operation would be $10,800 per day. This system, moreover, has not proved practicable. A plant using zinc oxide as a neutralizer would cost $3,000,000 and would require 500 tons of zinc-ore daily. Of the Cottrell method of deposition by static electricity, he said that its application was not practicable on a commercial scale, except under very peculiar conditions. He concluded that if the friction-system now used at Great Falls were successful, Anaconda would adopt it, though its installation might cost $2,000,000.

Mr. Mathewson's summary suggests one criticism: namely, by using the Fiechter asbestic-thread bags, with mechanical devices for shaking and collecting dust, the operating-cost of the bag-house would be made small and the life of the bags, once hung, indefinitely long.

II. History of the Condensation of Fume.

Notwithstanding the proved efficiency of the bag-house, and the opinion of competent metallurgists since 1860 that lead-fume must be filtered to insure recovery of values, costly attempts are still being made to secure this object by washing.

J. B. Wynne[1] has described an extremely thorough and eco-

[1] *Engineering and Mining Journal*, vol. lxxxviii, No. 13, pp. 602 to 604 (Sept. 25, 1909).

nomical method of washing gases, with the primary object of recovering the lead-, silver-, and antimony-fume. While Mr. Wynne was making his experiments, I was witnessing the installation of a washing-plant for the recovery of lead, bismuth, antimony, and silver carried in the fumes of an American smeltery. The results, apparently about the same in both cases, proved that by this method from 50 to 60 per cent. of the lead, from 80 to 90 per cent. of the antimony, and between 40 and 50 per cent. of the silver may be recovered. If the fume is attacked by moisture in the finest division as steam, and afterwards precipitated in scrubbing-towers or centrifugal washers, the result is the same. It is possible that a better recovery might be effected by the use of the Theisen washer; but there are few works in which that machine would be economical or would rival modern textile filters. Nor will Mr. Wynne's thorough methods of filter-pressing the sludge commend themselves in practice. It is strange that metallurgists did not give heed to the experiences of Dr. Percy's friend, who prophesied the superiority of filters over washing-devices. If they had done so, the efficient filtering-system known as the bag-house would have been introduced long ago. For the only sure and safe means of recovering silver- and lead-values in fume and flue-gases is the proper application of textile filters.

In recovering fume from copper-furnaces other methods take the place of the bag-house, and under certain conditions may be preferable. For instance, the system used at Great Falls, Mont., where a million Roesing wires, suspended in the flue-chamber and cooled in places by introduced air, collect much of the values contained in the gases, may be better suited to a plant smelting from 4,000 to 5,000 tons of copper-ore a day, in which the volatilization of silver is relatively smaller than in lead-silver smelteries.

On the other hand, in the special methods of Mansfeld, Germany, the employment, in internal-combustion engines, of gases of high calorific value from the blast-furnaces demands the thorough washing of the gas, and the recovery, in the Theisen washer, of the finest particles of fume.

In reviewing the history of metallurgy we are struck by the small progress and the lack of originality shown between 1780 and 1880 in the means devised for the condensation of fume.

In the majority of cases water-spraying was used. The result was not without value, since it developed the centrifugal scrubbers and the Theisen washer.

The first condensing-system seems to have been built in or about the year 1780.

In 1778 Bishop Watson published an essay on Derbyshire Lead-Ore, in which he pointed out that, during the smelting of galena, lead was sublimed in considerable quantity, of which part adhered to the internal surface of the chimney and part escaped into the air, to be deposited on the surrounding country, poisoning the water and herbage on which it settled. He proposed to collect the sublimed lead "by making it meet with water or with a vapor of water during its ascent, or by making it pass through a horizontal chimney of sufficient length." The first flue was built about this time at Middleton Dale, in Derbyshire, to prevent the injury of certain pastures by the smoke of the local lead-works. So much sublimed lead was deposited in this "horizontal chimney" that the practicability of saving at least a part of the volatilized metal was assured, and the wisdom of the Bishop's advice established.

We may compare with Bishop Watson's suggestions the observations on the condensation of lead-smoke by a "British Smelter," just 100 years afterwards. This gentleman had probably not read Bishop Watson's essay, yet the best means of condensation which all his experience and sense could suggest was a long flue, in conjunction with some possible textile filter not yet devised. He made experiments in filtering fume by passing it first into a condensing-chamber with water at the bottom to cool the smoke, thence forcing it to travel up and down partitions filled with small coke. The first smoke that passed the filters was white and rich in fume, but after a few hours the smoke became very much purer. In about four hours the appearance of the smoke showed that filtration was practically perfect. Because the draft in the furnace was stopped, the experiment was brought to a conclusion, the smoke diverted, and the top of the filter raised. "The contents presented a beautiful appearance. No coke was to be seen. It seemed as though snow had fallen lightly to the depth of an inch or so." This was, of course, lead-fume, which had formed a perfect and almost impermeable filter with the coke. After dis-

charging a shower of water over the coke a fan was set to work and more smoke drawn through the wet filter. This smoke contained much lead-fume. The shower was stopped and the process of filtration repeated until the draft ceased. When a continuous shower of water was used, certain passages were kept open, through which the smoke whistled, and condensation was ineffective. In all cases the dry coke formed a nearer perfect filter, and more quickly than the wet coke. He likewise found that dry flues favor precipitation rather than those containing water. These experiments showed that lead-fume in furnace-gas is, like a precipitate in water, capable of filtration; and the "British Smelter" suggested the use of a textile fabric as a filter, but feared the effect of the heat and acid gas on it. He also realized that agitation and friction favored the settling-action. These principles were later applied in such processes as the Cottrell, Roesing, and Freudenberg.

His condemnation of the empty chamber as a means of settling dust is very decided; and his remarks on the inefficiency of the steam-and-water condensing-method will find favor with all who have tried it. He declares rightly that it is easy enough to condense the fume-bearing steam to mist, but very difficult to get the mist to mix with or settle into water. He concludes that "nothing better for condensing has been proposed than the long flue, yet almost all who have had experience of flues would like to see an equal result attained with some other system of less compass and less cost in construction and maintainance."

This *résumé* of the "British Smelter's" experiences covers the greater part of the knowledge we have to-day. If we work in a civilized community we cannot escape the use of the long and expensive flue, terminating in a high stack. The Pertusola smeltery, for instance, tried filtering the fume directly behind the furnace-house; and although the texture of the asbestic-thread bags resisted the action of the heat, yet in the long run, on account of the ease of manipulation and the more-perfect recovery, it was found advisable to use the flue and adopt the American custom of placing the bag-house directly in front of the stack.

If we could only find some easy and profitable way of recovering the sulphurous and sulphuric acid there would be no

necessity for the high stack; for, since the draft is not increased by any additional height over 50 m., the cost of building above this point must be charged against the fume-account. It is true that our present flues are much better built and more solid and lasting than those of the "British Smelter." We have tried the Monier system, reinforced concrete, and solid and hollow concrete blocks in building flues, and abandoned them. The only condition under which a concrete flue has a chance of resisting the action of furnace-gases consists in a dry climate. We have done with flues made of steel plate, and have arrived at the conclusion that no other material can hold its own with brick and stone. True, that we must be extremely careful in the selection of mortar. Lime-mortar is out of the question. For copper-smelter flues, a bond of mud or clay must be used, while in the flues of the majority of lead-smelters, a mortar of hot tar and sand or fire-clay, and in special cases, as protection against very acid gases, a mixture of water-glass and asbestos, are found effective.

In constructing the brick flues, most Western smelters prefer building the walls thick and strong, thus foregoing the cooling-effect which would be obtained by making them thin and strongly buttressing them. At some works, little iron is used in the flues, the brick walls being retained by upright wooden posts held by tie-rods. But the more general method employs iron stay-beams and tie-bars. A specially strong construction is required for flues as large as those at Great Falls (48 ft. wide by 21 ft. high) and Anaconda (60 ft. wide and 20 ft. high).

On the whole, we may say that our progress since the time that the "British Smelter" wrote has consisted mainly in the development of methods of flue-building; the employment of duck, woolen, and asbestos-thread bags; and the installation, in more than an experimental way, of the zinc oxide method of neutralizing acid gases and the Cottrell method of precipitating fume and acid. We have also learned that apart from the smelting of iron and antimony and certain forms of copper-smelting, the spraying-methods are neither economical nor efficient. Finally, we have become convinced of the advantages of baffles in flues, and particularly in dust-chambers; and we expect to learn from the experiences of Great Falls the

benefit of combining the use of baffles with the admission of cold air at certain parts of the flue or dust-chambers to hasten the deposition of arsenic.

III. Description of Methods of Fume-Condensation.

The principal modern methods for condensing fume are:

1. Settling the dust and fume from the gases in a comparatively quiescent state, in chambers.

2. Cooling the gases by the admission of air, by radiation, and by spraying with cold water.

3. Recovery of fume by subjecting the gases to filtration or friction (such as that of bag-filters, baffles of metal sheets or wires hung in the flue), or by passing the gases through centrifugal washers.

4. Precipitation of fume by passing the gases through a vessel charged at numerous points with static electricity.

In regard to (1), Dr. Percy proclaimed the fact that, no matter how large the chamber, little deposition could be expected, unless it contained baffles of some kind. The general experience of metallurgists confirms this statement; and, the unbaffled dust-chamber is now restricted to the collection of solid particles which deposit immediately behind the furnace. The once-frequent comparison between the deposition of sediment from a river passing through a lake, and the supposed settling of fume from gases passing through a chamber, no longer arouses enthusiasm. In some years of experience with large chambers built near the stack-end of a flue, I found that a deposition of fume proportional to the greater cubic content of the chamber could be achieved only by placing in the chamber baffles, in the form of strip-iron, wires, or Freudenberg plates.

The chief methods employed in the cooling-systems of condensation marked (2), are:

(*a*) Water-spraying in showers or in be-sprinkled Glover towers; forcing the gases through a bath of water in the manner first practiced by Stagg; or passing them through centrifugal washers.

(*b*) Cooling the flues by circulating water about them; by presenting cool obstacles to the passage of the gases, such as permeable beds of coke, checker-work of bricks, metal wires

or plates suspended in the gas-current; by the introduction of cold air to the flue or dust-chambers; and finally by building flues of great length, and with thin walls, to secure cooling by radiation.

Of the many systems of wet-condensation, such as those of Pontgibaud, Stokoe, Stagg, Fallize, and Egleston, which are described in the treatises of Percy, Schnabel, and Hofman, I believe that scarcely one is now in use in its original form. I did, however, recently see a condenser similar in form to Stagg's, installed and operated at an American plant, with the usual poor results of such systems; the principle of the system of Egleston was followed at one time at Freiberg; and the flue cooled by outside water-circulation was still in use, in 1900, at Halsbruecken. The most complete system of wet-condensation which I know of in present practice is at the antimony-smeltery of the Société Anonyme Franco-Italienne, at Brioude. Here the water is sprayed directly against the blades of the fans which draw the gases from the furnace.[2]

The fume-condensing system of Tarnowitz probably embodies most of the features of the older systems. Briefly, the condensation is effected by driving and drawing the hot furnace-gases through a system of water-sprayed towers of the Glover type; spraying the fume in the flue before and after its passage through the various blowers; and settling and collecting the dust in *spitzkasten*. The wash-water is neutralized by pumping it through towers into which lime-water is injected. Thus almost all the sulphur which comes into the works finds its way to the dump in the form of sulphite and sulphate of lime—at a cost, however, which must seriously affect the economy of the process. The neutralization of the acid waters, which formerly ran off to the nearest stream, was insisted upon by the local forest-authorities.

In Germany the Theisen apparatus, so familiar to iron-smelters, has been very successfully introduced in the copper-works of Mansfeld. The Mansfeld apparatus thoroughly cleanses the furnace-gases of all solid material and permits them to be used as fuel for power, by ignition, either directly in gas-engines, or under boilers. The disadvantage of the

[2] *Mineral Industry*, vol. xvi., p. 61 (1907).

Theisen apparatus lies in the considerable horse-power required to drive it; its great advantage, in its thorough cleansing of the fume. In one works which I visited, it reduced the dust-content of the waste gases from 6 to 0.01 g. per cubic meter. It is the high calorific value of the Mansfeld gases, due to the large percentage of coke used in the blast-furnace, which makes the use of the Theisen scrubber economical.

Among the surface- or baffle-condensers (3), a notable invention was that of the Freudenberg plates, introduced at Ems early in the '80's by the superintendent after whom they were named. These plates received general application in the flues of German smelters, and are still used in several works. Of equal efficiency are the Roesing wires.[3] These were first used about 1890, and are still popular.

But an altogether new era was marked by the actual introduction and use by Bartlett of the textile-bag filter, now so widely used. And a still greater advance was made in 1902, when the Pertusola works entrusted to M. Louis Fiechter, of Basel, the difficult task of installing an apparatus for condensing and collecting the lead-fume, which should be as efficient as the dust-collecting devices which he had placed in many European works, and which had earned for him an international reputation. The outcome of Mr. Fiechter's work was the installation of a series of bags, or rather frames, of unwoven, specially prepared, asbestic threads, through which the dust was filtered. A mechanical shaking-device deposited the dust from the bags on the floor of the chamber, whence it was transported by screw-conveyors to bins or cars outside.

While the European metallurgists were developing the Fiechter system, American smelters were gradually becoming convinced of the importance and value of the Bartlett bag-house. In the early use of cotton bags, they had been discouraged by frequent disasters, due to the burning or corrosion of the fabric; and, at best, the bags lasted only eight or nine months. In order to lengthen the life of the fabric, resort was had to dipping in linseed oil or in fire-proof paint, but the bags thus treated lasted no longer than the original cotton bags. The life of the cotton bag now in general use is about 18

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[3] *Hofman's Metallurgy of Lead*, p. 287 (1892).

months—perhaps because more care is taken to cool the gases before they reach the bag-house, or, perhaps, because of the use, in the roasting-furnaces, of lime, which may neutralize the sulphuric acid produced. Woolen bags are also in use; and it is claimed by their advocates that their extra initial cost is offset by their longer duration.

In at least one American works, the disagreeable necessity of entering the bag-room has been removed by a mechanical arrangement permitting the bags to be shaken efficiently from the outside. It is probable that greater convenience and economy will be secured for the bag-house by the adoption of mechanical devices similar to those of the Fiechter system; and it is possible that, for the purpose of fume-condensation only, filtering through textile fabrics may win the day against any methods of filtering through water. Exceptions may be made of the antimony-smelter, who will use centrifugal scrubbers; the iron-smelter, who will employ the Theisen washer; and the copper-smelter, who will probably turn to the friction air-cooling method, and use Roesing wires or Freudenberg plates.

As for arsenic, it deposits completely in the bags of filter-houses. It was formerly considered a disadvantage of the bag-house system that it retained this fume; perhaps the pressure of the farmer's complaints may change this view. To condense arsenic completely, it is possible that some cooling-process, such as freezing, on the Gayley principle, may be developed. None of the water-cooling processes, such as spraying the gases or circulating water around the flues, will be found efficient for this purpose. Arsenic may be partly condensed by the radiation flue-system, in which the walls and roofs are made as thin as possible. One flue, recently constructed, was built half a brick thick and firmly buttressed, and seems to have been sufficiently strong. The method offers, however, no advantages over the baffled and air-cooled flue-and-chamber system.

The last-named system, in conjunction with the dust-chamber directly behind the furnace, and a bag-house before the stack, may become the accepted one for general metallurgical plants, particularly when silver and lead or zinc are important constituents of the charge. According to the results of my experiments, more silver is lost through the stack than the per-

centage in the dust at the bottom would indicate. In one instance, the proportion of silver to lead, in fume which was about to pass up the stack, was found to be much greater than that in the dust at the foot. It follows that a bag-house or filter of some kind should be used whenever silver is smelted.

For zinc-fume, condensation by radiation, air-cooling, and filtering through bags is effective, and as economical as any other satisfactory method. Mr. Pape[4] has described a method of utilizing the sensible heat in the gases from the Oker oxide furnaces, which pass from the dust-precipitation chamber to 12 boilers. The boiler-tubes cool the gases from 1,100° to 280° C. They then pass through economizers, or chambers of wrought iron. As they emerge from a smaller into a larger chamber, the flow is somewhat arrested and further precipitation is effected. Only the finest oxides now remain in the gases, which have a temperature of about 150° C., and, mixed with fresh air, are forced by fans into the bag-house, where the fine oxides are collected.

IV. The Neutralization of Furnace-Gases.

German metallurgists were early obliged by government regulation to roast all ores and products high in sulphur in furnaces connected with a sulphuric acid works. It was believed that the only sensible method of eliminating sulphurous acid lay in its conversion to sulphuric acid. Although this manufacture of acid is in itself an unprofitable industry for the smelter, yet it is possible that the enviable position which Germany now holds as a manufacturer and universal provider of chemical products, is in some measure due to the fact that large quantities of sulphuric acid were, at an early stage of the country's industrial career, available at very cheap rates for the manufacture of chemical products. I recollect that some years ago, German smelteries were selling chamber-acid at $4.50 per ton f. o. b. cars, while Freiberg's books probably never showed a profit from the manufacture of 66° B. arsenic-free acid.

Accordingly, metallurgists sought other economical means of eliminating or neutralizing the sulphurous acid contained in the fume. In 1880, Winkler, of Freiberg, suggested passing the fume through three contiguous chambers filled with limestone

[4] *Engineering and Mining Journal*, vol. lxxxix., No. 16, p. 820 (Apr. 16, 1910).

and covered with a perforated wooden roof. Water was sprayed through the roof upon the limestone, and the draft of the gases through the chambers was induced by a fan. The sulphurous acid was absorbed by the water; and the acid water was neutralized by the lime. Although the sulphurous acid content in the escaping gases was reduced by this means from 0.360 to 0.039 per cent. by volume, the process found only limited application at Tarnowitz and Freiberg.

Then Theodor Fleitmann proposed to pass the gases through a shaft-furnace filled with iron oxide and coke, with the entrance of sufficient air to burn the fuel. The oxides of sulphur were to be reduced and the iron sulphide collected at the bottom of the furnace. This method probably never passed the experimental stage.

A distinct advance was the invention, in 1881, by Professor Schnabel, of a rational method of absorbing sulphurous acid.[5] Zinc oxide is moistened and distributed through a quantity of filter-material, such as heather. On drawing the furnace-gases through this filter the zinc oxide combines with the sulphur dioxide, forming zinc sulphite, from which it was proposed to regenerate the zinc oxide by heating or roasting the sulphite, preferably in an acid manufactory. F. M. Lyte[6] uses in a similar way iron hydrate and zinc oxide.

H. Precht[7] made the interesting suggestion in 1882 of using alumina and magnesia hydroxides as absorbents.

A modification of Schnabel's method has been applied at the plant of the United States Smelting Co. in Utah. Zinc oxide in the form of dust is used on the bags or distributed in the form of fume in the gases to neutralize and collect the sulphuric acid contained in the smeltery-gases which filter through the bags.

The Tennessee Copper Co. is to be congratulated on the successful application of blast-furnace gases in the direct manufacture on a large scale of sulphuric acid by the chamber-process.

To a metallurgist educated in the German school, the very favorable and profitable agreement made between the smelter

[5] German patent, No. 16,960 of 1881.
[6] English patent, No. 5,416 of 1881.
[7] German patent, No. 17,000.

and the superphosphate factory betokens possibilities, hitherto regarded as incredible, of combining the successful elimination of the sulphurous acid gases in the fume with a profitable marketing of the acid, in the form of sulphuric. We have seen in Germany the profit arising to farmers from the use of superphosphate. It seems that the compact recently made in Tennessee marks a new stage in domestic economics.

In recent discussions of means for overcoming the acid-fume nuisance, one eminent metallurgist was of the opinion that the only radical way in which the Salt Lake, the California and other smelters could meet the complaints of the farmers was by driving the fume with fans through long flues to outlets in barren regions where neither forest nor farm could be hurt. This does not seem to mark a great advance on Bishop Watson's method of a long horizontal chimney, proposed 130 years ago.

It has been thought possible that a success scarcely less important than that of Mr. Gayley, in freeing the iron furnace-blast of water-vapor, by condensing the latter in solid form at a low temperature, might be achieved by some enterprising metallurgist who should apply similar principles to the elimination of sulphurous acid from fume. So far, however, no experimental results, which would warrant the installation of a freezing-apparatus in a large plant, have been obtained.

It has also been proposed from time to time to settle fume and dust, and to precipitate acids contained in the furnace-gases, by charging these gases with static electricity from a number of points on a cylinder or vessel introduced into the flue or chamber, so that the fume-dust and aqueous acids might be repulsed from such points, and deposited on the surfaces ot opposite poles, which may be the floor or sides of the flue or chamber receiving the gases. A method embodying these principles has been tried at the Selby works under the name of the Cottrell process.[8]

Perhaps the most efficient method at present in use for condensing arsenic, in plants which do not use the bag-house or some similar textile filtering-system, is the Great Falls system, of large flues with a chamber hung with a myriad of wires. The

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[8] *Engineering and Mining Journal*, vol. lxxxvi., No. 8, p. 375 (Aug. 22, 1908).

deposition of the condensable arsenic in cooler parts of the chamber is helped by introducing air, while the mechanically-carried dust-particles are caught on wires in the hotter part, behind the entrance of the flues from the smeltery. Accordingly, the wires are hung in the chamber in two batteries, with an intervening vacant space, into which ducts from the basement and the roof lead the cooling air.[9]

A peculiar instance of the revival of the old reduction-method of freeing furnace-gases from SO_2 by passing them through a reducing atmosphere is contained in the proposals of F. R. Carpenter,[10] which cover the following methods:

1. Separation of SO_2 by freezing and subsequent recovery of S by reduction.

2. Passing the gases through a reducing atmosphere and recovering the reduced S by settling or filtering.

3. Converting part of the SO_2 into H_2S and producing S by the interaction of the two gases.

4. Converting the whole of the SO_2 into H_2S ($2SO_2 + 2H_2O + 3C = 2H_2S + 3CO_2$; $SO_2 + H_2O + 2C = H_2S + CO_2 + CO$; $SO_2 + H_2O + 3C = H_2S + 3CO$) and producing S from this by means of the Clause-Chance kiln: $H_2S + O$ (from Fe_2O_3) $= S + H_2O$.

I know of no publication of the results of experiments which may have been made with this process on a practical scale.

V. A Résumé of Methods of Treating Furnace-Gases Applicable in Modern Practice.

1. *Iron-Blast-Furnace Gases.*—Dust collected in separators and Theisen centrifugal washer. Thermal values realized in heat-economizers and, after cleaning, by combustion in gas-engines.

2. *Open-Hearth Steel-Furnace Gases.*—These contain no dust to speak of. The thermal values are realized by using the regenerative-chamber system or using the waste gases to heat boilers.

3. *Copper-Furnace Gases.*—Dust deposited .in the long flue-and-chamber system, with baffles. All the copper recovered and most of the arsenic. It is questionable how much silver

[9] *Engineering and Mining Journal*, vol. lxxxix., No. 7, p. 368 (Feb. 12, 1910).

[10] United States patent, No. 871,912, Nov. 26, 1907.

is recovered by this method. The thermal values of the reverberatory-furnace waste gases are generally realized in the same manner as those from the open-hearth.

4. *Lead-Silver-Furnace Gases.*—The best system suggested is the chamber behind the furnace, with baffles or partitions, to offer a surface to the gases and aid precipitation; a long flue, with or without baffles, with walls favoring radiation, connecting with the dust-chamber at the furnace and with a sufficiently large bag-house before the stack: the bags to be shaken and the dust mechanically removed. While cotton duck and woolen fabric as materials have their advocates, I think that the Fiechter asbestic-thread bag is most economical and efficient.

5. *Antimony-Furnace Gases.*—To the antimony-smelter there can be little question of choice: for the centrifugal scrubber has proved extremely efficient in recovering antimony-fume. It is possible that the tin-smelter would find this system of equal service in his work.

Of the other methods proposed for recovering the sulphurous acid for rendering the gases harmless, it is difficult to realize that any can be used successfully in the plants which produce such quantities of gas as the larger Western smelteries make every day.

The zinc-oxide neutralization may secure a limited application when the question is one of preventing the escape of sulphuric acid. For the freezing method it is difficult to predict a future. The same might be said of the Cottrell method. It is conceivable, however, that the problem will be solved in good time by the production of a market for sulphuric acid at a future date through the industrial development of the Western States. We may then look for the application of a modification of the process used by the Tennessee Copper Co.

To recover the arsenic and silver from the fume and gases of copper-smelters, it is still possible that some such filtering method as the Fiechter or other bag-house system may be generally installed, with perhaps the use of zinc oxide in the form of fume in the flues, or as dust on the bags, to collect the sulphuric acid. The United States Smelting Co., at least, is planning some such installation in its plant at Bingham Junction, Utah. The Great Falls engineers expect a large proportion of the

arsenic-fume to condense on the wire baffles, particularly when assisted by the introduction of air from above and below the flue-chamber. Since the practice of the United States Smelting Co., in its present development, is scarcely applicable in very large smelteries, we shall regard the Great Falls efforts with the deepest interest and sympathy.

Manganese-Ore In Unusual Form.

BY WILLIAM P. BLAKE, TUCSON, ARIZ.*

(Canal Zone Meeting, November, 1910.)

A DEPOSIT of manganese-ore near Tucson, Ariz., merits notice by reason of the peculiar form in which it occurs, and as a striking example of ore-deposition by vadose upward capillary flow of metal-bearing solutions with concentration at the surface of the country-rock.

This deposit occurs at the northern end of the Coyote mountains, about 40 miles west of Tucson, upon the outcropping beds of porphyritic rock, which appear to be partly-metamorphosed strata of volcanic tufa, probably of late Secondary or of Tertiary age. The strata dip northward at a low angle, and present a series of basset-edges in parallel lines of outcrop, east and west, facing the south and extending over many acres.

Successive groups of these outcrops, from 5 to 20 ft. in breadth, are coated with manganese oxides in lustrous black films and crusts, shining in the sunlight like polished ebony, and covering the whole surface of the rock above the soil.

Seen at a little distance, these croppings look like solid ledges of massive and pure ore; but closer inspection and a few blows of the hammer reveal their true nature as a superficial coating allied in origin to the blackened shining surface of rocks exposed in desert regions, familiarly known as " desert

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* By reason of the death of the distinguished author, this communication, probably his last communication to technical literature, is published without revision of the proofs by him.

paint" or "desert varnish,"[1] similar to the phenomena described by me in a former paper,[2] but much thicker and heavier.

The deposit of manganiferous oxides varies in thickness from a mere film, not thicker than ordinary paper, to crusts 2 or 3 in. thick and varying, in the weight of detached masses, from a few ounces to many pounds. Such heavy masses may be broken from the croppings, or, having been detached by weathering, may be found lying in the soil.

The ore not only exists in these crusts, but also permeates, in greater or less degree, and in some places traverses in ramifying seams, the upper portion of the rock under the crusts. Masses broken from the croppings may thus have a sponge-like skeleton of rocky substance, which in analyses appears as insoluble matter.

The approximate richness of the ore in manganese, on a commercial scale, was ascertained by means of a general sample composed of fragments, broken from the croppings over a considerable area, and then divided into two classes according to its specific gravity as compared with that of the barren country-rock. Assays of these two classes resulted as follows:

No. 1, *Best Ore.*—Manganese, 53 per cent.; insoluble, 12 per cent.

No. 2, *Second-Class Ore.*—Manganese, 12 per cent.; insoluble, 58 per cent.

Iron and silica were found to be abundant in each sample. Some silica appears to be in combination, and suggests the presence of braunite; but the massive ore is a very hard, black, non-crystalline mixture of psilomelane and manganite.

There is no direct evidence of the source of the manganiferous solutions. The occurrence at Tombstone, in the adjoining county, of large masses of the sulphide (alabandite) in the Lucky Cuss mine,[3] where there was much black manganese-ore in crusts below the surface, suggests that this sulphide may be disseminated in depth in the strata below the Coyote deposit;

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[1] A term first given by G. K. Gilbert. See *Rocks, Rock-Weathering, and Soils*, by George P. Merrill, p. 256 (New York, 1897).

[2] Superficial Blackening and Discoloration of Rocks, Especially in Desert Regions, *Trans.*, xxxv., 371 (1905).

[3] See my paper, Tombstone and Its Mines, *Trans.*, xxxiv., 670 (1904).

but there have been no excavations deep enough to determine this point. In the silver- and copper-bearing veins of the Salero mines, in Santa Cruz county, 70 miles to the east, manganese-spar is common as an associated mineral in the veins which traverse diorite, and gives, by its decomposition, an abundance of black manganese-ore in the croppings. The association of manganiferous minerals with silver-ores is well known, not only in Arizona but in Montana and elsewhere. At the Alice and other mines near Butte, Mont., for example, the croppings of the veins may be traced by the black manganese oxide in the soil. This common association suggests the possibility that, at the Coyote mountains "manganese-field," explorations in depth may develop ores of silver and other metals.

The abundance of the crusts and their localization upon the croppings of the clastic rocks widely separate the Coyote deposits from the thin, almost imperceptible, coatings of the desert-rocks generally. I have found the latter, however, to vary greatly in thickness, being, in this respect, evidently dependent upon the nature and abundance of the subterranean supply. It seems hardly possible that the decomposition of mica or other silicates (unless silicates of manganese) could furnish enough manganese to form such heavy coatings as I have here described. The phenomena of the ordinary discoloration of rock-surfaces by manganese oxides are well described and their origin is ably discussed by George P. Merrill in his admirable work, *Rocks, Rock-Weathering, and Soils*, already cited.

I offer this brief communication for the purpose of calling attention to a class of superficial manganiferous deposits which seems to me to require the hypothesis stated in my introductory sentence.

The Laws of Intrusion.

BY BLAMEY STEVENS, VALDEZ, ALASKA.

(Canal Zone Meeting November, 1910.)

I. Introduction.

The object of this paper is to show how igneous intrusion is governed by definite mechanical laws. A distinction is made between dikes and fissures, and the various characteristics of intrusions are explained on an exact scientific basis. The modification of rock-stresses by intrusion is then considered and extensive isostress regions are recognized. Finally, a complete classification of intrusions is constructed covering all possible cases.

The generalizations of observed data, taken from Geikie's *Text-Book of Geology*, conform to the laws of intrusion herein stated, and the deductions based upon these laws.

No authority, however, is quoted for the type which I have named a corrugated intrusion. Its existence is a necessary deduction from the laws of intrusion, and even the entire absence of examples in technical publications cannot disprove its reality. In all probability, corrugated fissures would have remained undiscovered, but for the working of mineral veins. As intrusions of tabular form are seldom worked for commercial purposes, it has been left for theory to anticipate the discovery of corrugated dikes.

In the light of these laws of intrusion and the corresponding laws of fissures,[1] there seems to be a remarkable uniformity of stress-distribution in rocks at moderate depths. This renders possible the mathematical calculation which should help us to approximate exact science in this and kindred branches of applied geology.

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[1] *Trans.*, xl., 475 (1910).

II. Former Theory.

The formation of intrusions does not seem to have been previously investigated by any exact method. Thus in Geikie's *Text-Book of Geology*[2] we find the following vague statement of a law of intrusion:

"The general law which has governed the intrusion of igneous rock within the earth's crust may be thus stated: Every fluid mass impelled upwards by pressure from below, or by the expansion of its own imprisoned vapour, has sought egress along the line of least resistance. That line has depended in each case upon the structure of the terrestrial crust and the energy of eruption. It may have been determined by an already existent dislocation, by planes of stratification, by the surface of junction of two unconformable formations, by contemporaneously formed cracks, or by other more complex lines of weakness. . . . The shape of the channel of escape has thus determined the external form of the intrusive mass, as the mould regulates the form assumed by cast-iron."

This statement seems to indicate a general notion that dikes are formed like fissures. But the same author notes several differences between dikes and veins. Thus:

"Dykes differ from veins in the greater parallelism of their sides, their verticality, and their greater regularity of breadth and persistence of direction." (*Id.*, p. 583.)

"It might be supposed that necks should always rise on lines of fissure. But in central Scotland, where they abound in rocks of Carboniferous age, it is quite exceptional to find one placed on a fault." (*Id.*, p. 585.)

"The ascending molten matter [of intrusive sheets], after breaking across the rocks, or rather, after ascending through fissures, either previously formed or opened at the time of the outburst, has at last found its path of least resistance to lie along the bedding-planes of the strata." (*Id.*, p. 573.)

III. Author's Theory.

I propose to show in this paper that the main factor governing igneous intrusion is the system of stress to which the rocks are subject; that existing dislocations, planes of stratification and other accidents and complex lines of weakness are of secondary importance in determining the shape and position of the intrusion.

Varieties of Intrusion.

Intrusions may be divided into two general forms, viz.: regular, formed in what may be called fissuring-country; and irregular intrusions, formed in what may be called equi-pressure country.

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[2] Book IV., Part VII., Sec. i., p. 564, 3d ed. (1893).

As shown in my paper on The Laws of Fissures,[3] the ratio of vertical to horizontal stress in fissuring-country is considerable, being five to one when the coefficient of friction is 0.9. In considering intrusions, the water-pressure has to be taken into account, since this also has to be balanced by intrusive pressure. The ratio is then reduced to about two to one in normal fissuring-country. In Figs. 1 to 3, this ratio is exaggerated to 3 to 1 in order better to illustrate the differences between various combinations of stress.

In equi-pressure country the rock-pressures are about equal in all directions. It will be shown later that equi-pressure country may be caused by magmas intruding into normal fissuring-country.

There is in general a fair distinction between the regular and irregular varieties, the intermediate forms not being very common.

IV. Laws of Intrusions.

The following laws hold for all regular intrusions: and, with some extra definitions of terms, to be given later, they may also be said to hold for irregular intrusions:

1. The break, or orifice, filled by an intrusion is formed by the pressure of the intruding magma at the time of intrusion and not far distant from the front of the intruding magma.

2. The plane of a tabular intrusion is perpendicular to the direction of least stress of the rock into which the magma is intruding.

3. When there is more than one direction of least stress a neck may be formed, the axis of which corresponds with the direction of greatest stress.

V. Detailed Analysis.

Tabular Intrusions.

The condition of stress in any one place in a rock-mass or other solid substance may be completely represented by three principal stresses (p_x, p_y, p_z) at right angles with one another. Stresses in intermediate directions have intermediate values; they are not simple or direct, but have a tangential component. The cohesive stresses of the material may be defined as the

[3] *Trans.*, xl., 478 (1910).

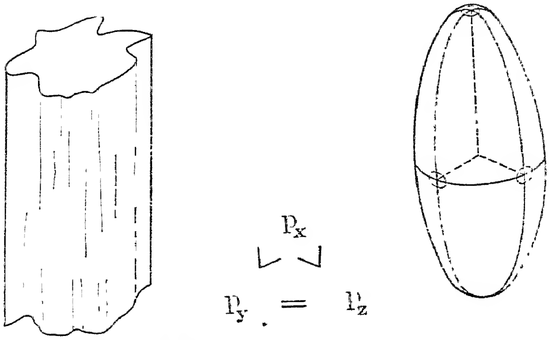


FIG. 1.—LINEAR INTRUSION (NECK).

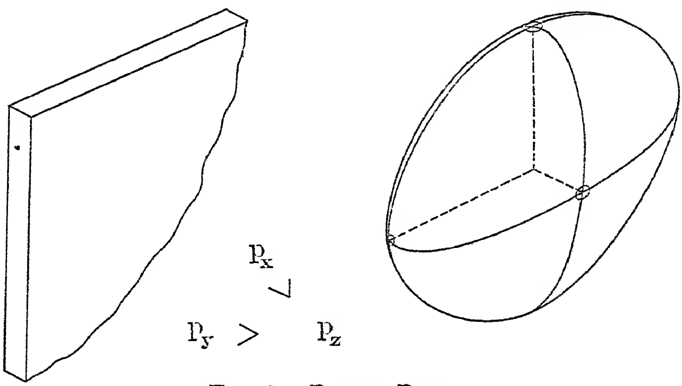


FIG. 2.—PLANE DIKE.

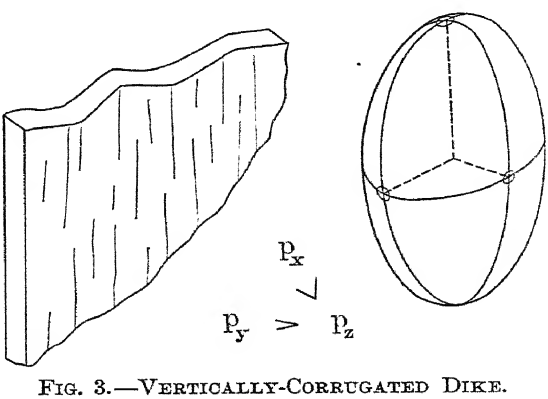


FIG. 3.—VERTICALLY-CORRUGATED DIKE.

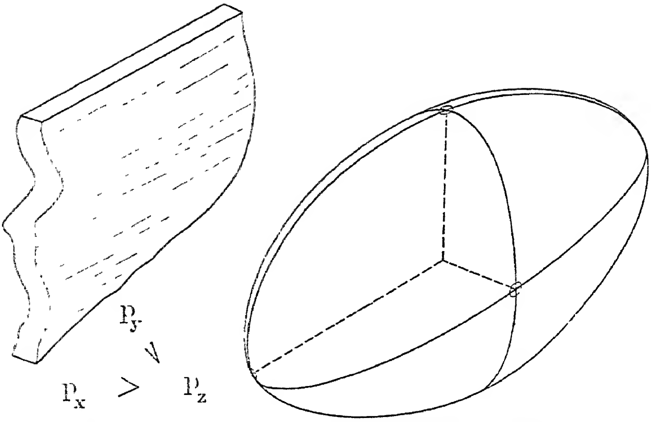


FIG. 4.—HORIZONTALLY-CORRUGATED DIKE.

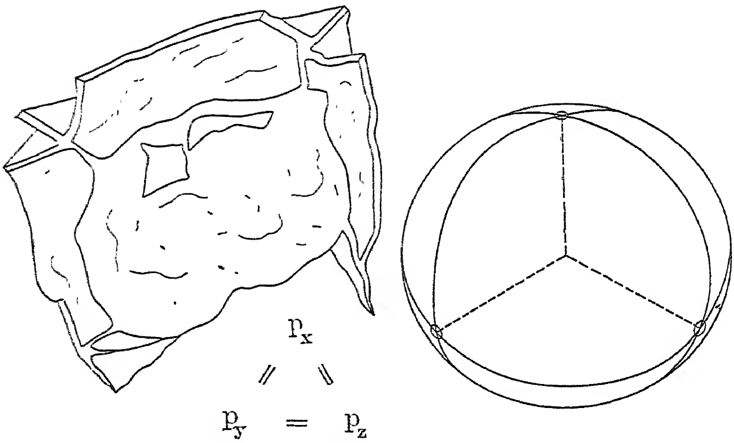


FIG. 5.—SKEW INTRUSION.

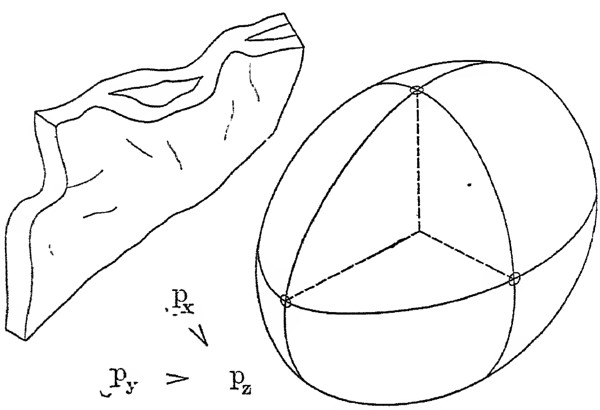


FIG. 6.—IRREGULAR DIKE.

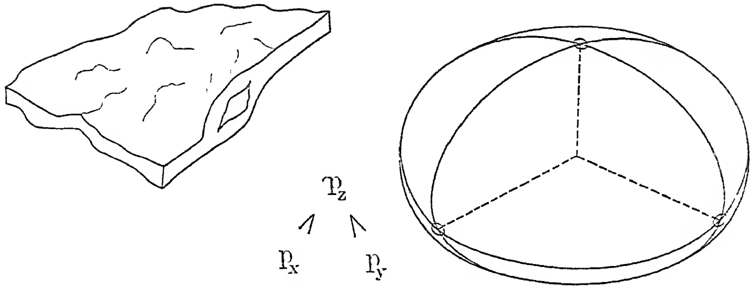


FIG. 7.—INTRUSIVE SHEET (SILL, LACCOLITH).

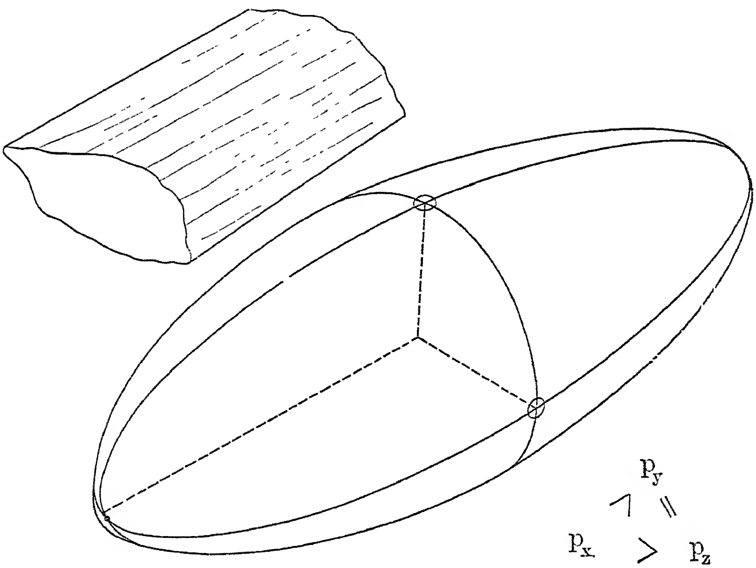


FIG. 8.—HORIZONTAL NECK (RARE).

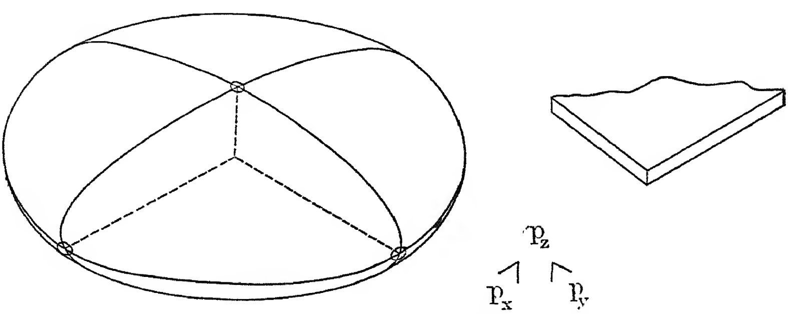


FIG. 9.—PLANE INTRUSIVE SHEET (RARE).

tensions necessary to rupture the material across all directions when it is not buried. These cohesive stresses may be added to the rectangular stresses, so as to apply the laws with exactitude when the breaking asunder of any cleaved or jointed rock is considered; but when the cohesive-stress values are small, compared with the stress-differences at the great depths considered, the cohesive component may be neglected.

As Geikie says, the intrusive magma will evidently follow the path of least resistance. Expressed mathematically, this path is the channel in which the greatest amount of magma may accumulate with the least amount of work performed on the rock into which it intrudes. Therefore, if b be the mean breadth of a dike, Δ the area of each wall, and p the mean pressure of the rock at right angles to it: the work done in

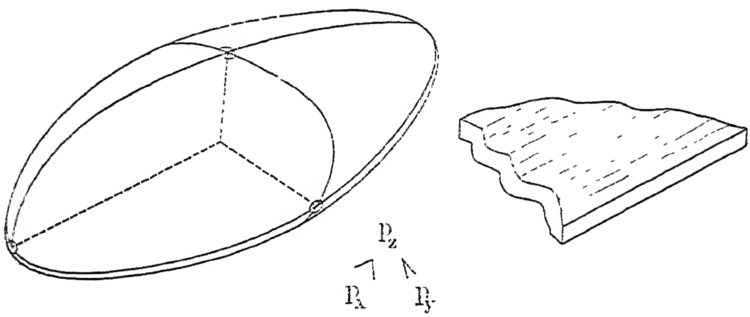


FIG. 10.—CORRUGATED INTRUSIVE SHEET (RARE).

separating to the distance b the two walls is $pb\Delta$ and the amount of magma stowed away is $b\Delta$, so that the work per unit cube of magma is p. If p_z is the least stress, the principle of least work will best be satisfied when the dike is perpendicular to it, *i. e.*, in the plane containing p_x and p_y. This is the second law. (See Figs. 2, 3, 4, and 6.)

Linear or Neck-Intrusions.

On the other hand, if p_z be equal to p_y, and both be less than p_x, a dike is equally likely to form perpendicular to the stress p_y as to p_z. In either case, however, its direction would contain the stress p_x and it seems natural to expect the formation of a linear intrusion including or parallel to this stress. Probably some small dike in a random direction is first formed,

say in the plane perpendicular to p_z. Owing to the elasticity of the rock, the stress p_z is thus locally increased so that it becomes slightly larger than p_y. Another small dike is then formed perpendicular to the stress p_y. This intersects the former dike along a line corresponding with the stress p_x (Fig. 11). The rock-stress in every direction radiating from this line is thus increased. The lateral elasticity of the material soon distributes this stress in perpendicular directions. If Fig. 11 and Fig. 12 are two cross-sections taken perpendicular to p_x, Fig. 11 being taken further up from the intrusive supply than Fig. 12, it is easy to see that those stresses which radiate from the line of p_x are smaller in Fig. 11 than in Fig. 12. The intrusive magma therefore finds its path of least resistance in enlarging the cross-dikes of Fig. 11 to the size of those of Fig. 12, and the intrusion thus follows up along in a direction which is coincident with the axis of greatest stress. This is the third law.

In actuality we might have three or more meeting but not intersecting dikes parallel to p_x, as in Fig. 13, or a series of dikes forming the sides of a vertical prism. In any case the internal corners will ultimately be broken off or dissolved and the system of rents will become the more or less circular orifice which forms a volcanic neck (Fig. 1).

Geikie describes a neck shape as follows:

"A neck is circular or elliptical in ground-plan, but occasionally more irregular and branching, and may vary in diameter from a few yards up to two miles, or even more. It descends into the earth perpendicularly to the stratification of the formation with which it is chronologically connected. Should rocks originally horizontal be subsequently tilted, a neck associated with them would of course be thrown out of the vertical." (*Op. cit.*, p. 585.)

Intrusive Sheets (Sills and Laccoliths).

An intrusive sheet differs from a dike only in the position of the axis of least stress (p_z). If this is horizontal, a dike is formed, if vertical, an intrusive sheet. (See Figs. 7, 9, and 10.) Intrusive sheet is a general term for any intrusion whose longer dimension is approximately horizontal. A sill is a thin intrusive sheet, and a laccolith is a thick one.

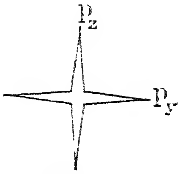


Fig. 11.

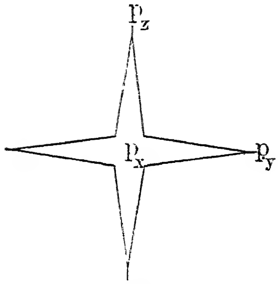


Fig. 12.

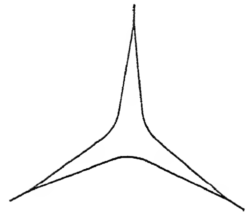
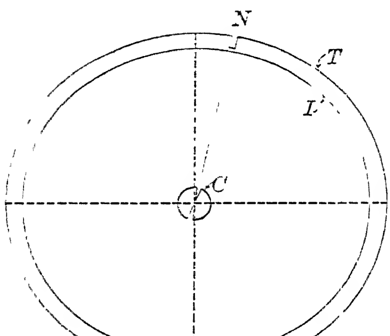


Fig. 13.

Figs. 11 to 13.—Formation of Linear Intrusions.



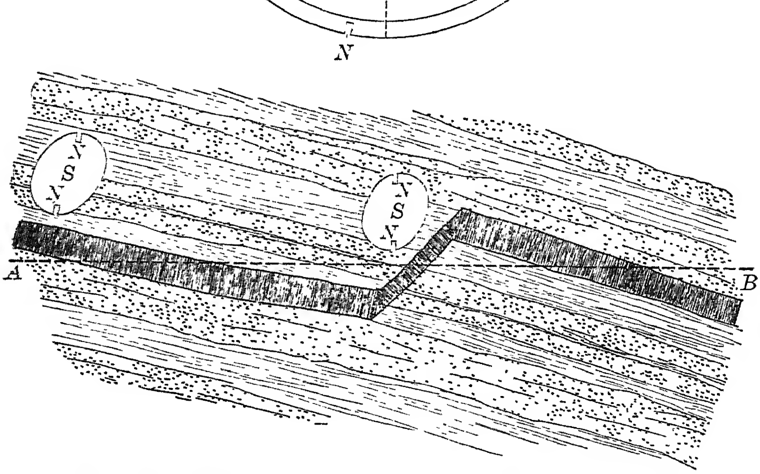


Fig. 14.—Irregular Intrusive Sheet (Sill, Laccolith).

Directions of Stresses.

In my paper on The Laws of Fissures,[4] I have shown that, more or less approximately:

1. The mean vertical stress over any considerable area is equal to the weight of the overlying rocks.

2. This being a principal stress, the other principal stresses are horizontal.

3. The mean inclination of a fissure to the least principal stress at the time of formation is about 66° (when the coefficient of friction is 0.9).

Directions of Intrusions.

These being the stresses at the time of an intrusion, we see at once that where cleavage and jointing do not interfere:

a. A dike should be vertical at the time of intrusion.

b. An intrusive sheet should be horizontal.

c. A neck should be vertical or horizontal.

d. The inclination of a fissure to a contemporaneous dike, neck, or intrusive sheet is about 24° (the coefficient of friction being 0.9).

Diagrammatic Representation.

The several kinds of intrusion are shown diagrammatically in Figs. 1 to 10, inclusive, which are isometric projections of the intrusions alongside the corresponding ellipsoids of stress, the principal axes of which are equal to the principal stresses. In the accompanying symbolic triangles the symbols p_x, p_y, and p_z represent the three principal stresses, the one at the apex of the triangle representing the vertical stress and the others the horizontal stresses.

The symbols represent:

$=$, "equal to;"
$>$, "slightly greater than;"
$<$, "slightly less than;"
$>$, "much greater than;"
$<$, "much less than."

The vertical stress is made the same in several of the figures, as it is strictly proportional to the depth. Where a relation

[4] *Trans.*, xl., 478 to 484 (1910).

between two principal stresses is not of importance, it is not indicated. The interpretation of the symbolic triangle may be illustrated by the vertical-neck triangle $\begin{array}{c} p_x \\ \diagup \quad \diagdown \\ p_y = p^z \end{array}$ which represents that in a vertical neck the principal stresses p_y and p_z are horizontal and equal to one another, and that each of these stresses is much less than the vertical stress, which in this case is p_x.

Subsequent Change of Direction.

As suggested by Geikie, the dip and pitch, and perhaps the strike, of a dike, fissure, or other corrugation or shoot may be considerably altered after its formation by the general tilting of the inclosing rocks. Although this is self-evident, it needs to be emphasized, since it is too little regarded by engineers and geologists.

Plane Dikes (Fig. 2).

If a dike should follow any other path in the formation than one perpendicular to a principal stress, a tangential stress would be introduced. This could not be sustained by the liquid magma filling the dike, as liquids can only sustain a direct stress. Therefore any extended intrusion formed under conditions of great stress-difference would have walls which are nearly perfect planes. Such intrusions may be called plane dikes.

Corrugated Intrusions (Figs. 3, 4, and 10).

On the other hand, it may happen that the two least principal stresses (p_y and p_z) may so nearly approach one another in size that the tangential stress, along all planes containing the direction of greatest principal stress (p_x), is very small. If the rock-formation is such as to induce it, intrusions formed under these conditions will have marked irregularities shown on a cross-section taken perpendicularly to the stress p_x, but will have a plane linear form shown on a cross-section taken perpendicularly to the stress p_y (p_z being slightly less than p_y). These intrusions may be called corrugated, and are of three kinds: vertically-corrugated dikes (Fig. 3); horizontally-corrugated dikes (Fig. 4); and corrugated intrusive sheets (Fig. 10).

Of these, by far the most important is the vertically-corrugated dike; the corrugated intrusive sheet is probably very rare. Necks may be considered as special cases of corrugated intrusions.

Irregular Intrusions.

We now come to a series of cases in which all three of the principal stresses are nearly equal. We shall call the stresses as induced by gravity and other conditions external to the cohesive properties of the rock, the induced stresses, to distinguish them from the cohesive stresses which give the rock its resistance to rupture by tension. The sums of the induced stresses and the cohesive stresses we shall call the total stresses, and it is these only which we have so far considered.

Graphical Representation of Stress-Systems.

A system of induced stress is always represented by an ellipsoid, the principal axes of which correspond in length and direction with the size and direction of the principal stresses. The corresponding diagram of cohesive stress may be any irregular figure, and the total stress is in this case also an irregular figure. For two-dimensional illustrations the ellipse is used instead of an ellipsoid. In Figs. 14 and 15, which represent stratified rocks, I is the ellipse of induced stress, C the figure of cohesive stress, and T that of total stress.

If we consider the breaking of a rent in the rocks, we find that its direction is determined by the total stresses. When it becomes filled with liquid magma, however, the cohesive element of the stress is no longer active, and if the rent is not parallel to an axis of induced stress, a tangential stress will be continually accumulated in front of the rent which is being broken in the solid rock (see Fig. 14). This tangential stress will locally alter the ellipse of total stress, so that at S (Fig. 14) it is like the small-scale ellipse drawn around the letter S. A new rent will then start at right angles to the shorter axis of this ellipse, the stress represented by this axis having become slightly smaller than the stress at right angles to the stratification. The latter is represented by the direct component of the stress measured between the bottoms of the notches, N, N.

It is important to notice that the mean path of the intrusion

is a line, AB, perpendicular to the axis of least induced stress, so that the second law of intrusion applies to the mean or local paths, according as the induced or total stresses, respectively, are considered. It will be seen that it is barely possible that the mean path of the intrusion might be perpendicular to one of the other axes of induced stress.

Skew Intrusions.

If the axes of principal induced stresses are perfectly equal, the path of intrusions will be entirely governed by the cohesive stresses. It will not be any more affected by the induced stress than the equilibrium of a small body on the surface of the earth is affected by the comparatively great pressures of the atmosphere.

Fig. 15 shows how the relatively small cohesive stresses

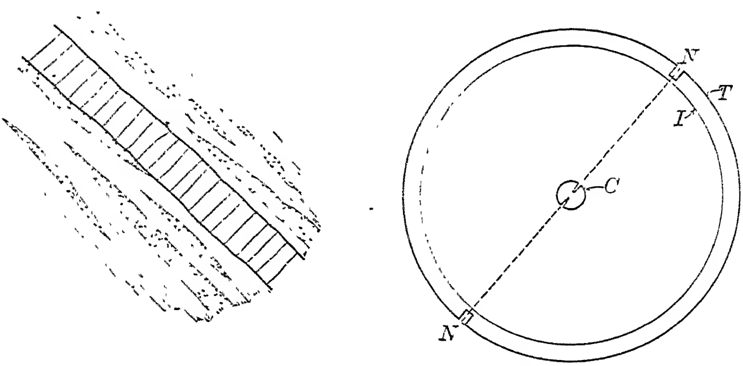


FIG. 15.—SKEW INTRUSION (BEDDED).

may thus govern the direction of an intrusion. Such intrusions may be called skew intrusions, and the particular case illustrated by Fig. 15 might be called a skew bedded intrusion. We can also expect to see skew fissure intrusions, and we might call the intrusion of Fig. 5 a skew ramifying intrusion. The extreme difference in shape of the intrusions of Figs. 5 and 15 is entirely due to the character of the formation and not to any radically different systems of induced stress.

Irregular Sills (Figs. 7 and 14).

If the horizontal induced stresses are slightly greater than the vertical induced stress, the mean plane of the intrusion is

horizontal and is represented by AB of Fig. 14. In this figure the formation is shown as tilted, and while the intrusion follows this plane for some distance, it has occasionally to cut back across the formation in order to get rid of the accumulating tangential stress. Geikie says of irregular sills:

"They do not rigidly conform to the bedding of the rocks among which they are intercalated, but sometimes break across it and run along on another platform. They catch up and involve portions of the surrounding strata." (*Op. cit.*, p. 573.)

As in other cases, the induced stress should determine the name of the intrusion, and the character of the formation may be tacked on to describe it more specifically. We might thus have irregular bedded sills (Fig. 14) or irregular ramifying sills (Fig. 7).

Irregular Dikes.

If in a system of nearly equal stresses, the least stress is horizontal, a dike is formed, the mean plane of which is vertical. Such a dike may in part follow the irregular jointing of the formation, in which case it will be called a ramifying dike, or it may in part follow some more regular vertical plane of weakness. There might thus be cleavage-dikes, fissure-dikes, bedded dikes, etc. Geikie, speaking of the latter, says:

"Sometimes they run for many yards or miles in tolerably straight lines. When this takes place along vertical or highly-inclined stratification, they look like beds, but they are of course really intrusive sheets. They may frequently be found to break across the bedding in a very irregular manner." (*Op. cit.*, p. 578.)

Original Intrusions.

From The Laws of Fissures we learn that the ratios of greatest to least stress are high in most rocks of moderate depth which remain undisturbed by intrusion. The original intrusions of a district are therefore necks, and plane and corrugated dikes and sills (Figs. 1, 2, 3, 4, 8, 9, and 10).

Intrusive Stresses.

The effect of an intrusion is to increase the horizontal stresses which give rise to necks and dikes. The rock-stresses of whole regions may thus be modified by a system of dikes. As previously stated, when a dike is rent open the horizontal

stress is smaller than the vertical. When the dike has broken through to the surface, however, there is a liquid column of magma whose pressure is about equal to the vertical pressure —or greater, if the viscosity of the magma is taken into consideration. This being a liquid, the horizontal stresses are equal to the vertical, and, after the dike has been widened so that the horizontal rock-stresses balance the magma-pressure, consolidation of the magma may take place. With further magma-supply, a sill, laccolith, or stock may be formed, according to the fluidity of the magma, the depth of the intrusion below the surface, the plane of weakness of the formation, etc.

Secondary Intrusions.

The intrusions thus formed may be called secondary intrusions. If the intermediate stress (p_y) is horizontal, it will be exceeded by the magma-pressure before the magma-column reaches the surface. In this case dikes making an angle with the main dike may be formed; these will be vertical corrugated dikes, verging into irregular dikes or sills and laccoliths as the magma-head increases.

Vertical necks may also be modified as above explained, but an intrusion which starts as a sill or intrusive sheet can only be modified to form a laccolith, and cannot at any considerable depth give place to other forms of intrusion except on a localized or sub-secondary scale.

Laccoliths are usually secondary intrusions. Geikie says of them:

> "They are connected with dykes or pipes which, descending through the rocks underneath, have been the channels by which the sheets were supplied." (*Op. cit.*, p. 573.)

Bosses (Figs. 5, 6, 7, 16, and 17).

Bosses may be either primary or secondary intrusions. Geikie (p. 564) defines them as amorphous masses consisting chiefly of crystalline coarse-textured rocks, granite and syenite being the most conspicuous examples.

No rigid classification of bosses has been attempted, owing, no doubt, to their irregular outlines. It is generally agreed that their shapes have been modified by their having eaten away the surrounding rocks; thus Geikie says further:

"On a smaller scale usually than granite, other crystalline rocks assume the condition of amorphous bosses. Diorite, syenite, quartz-porphyry, gabbro, and members of the diabase and basalt family have often been erupted in irregular masses, partly along fissures, partly along the bedding, but often involving and apparently melting up portions of the rocks through which they have made their way." (*Op. cit.*, p. 571.)

Boss is a general term for a large thick intrusion. The boundaries of bosses are often so modified or obscured from view that no more definite name can be applied. A laccolith has been previously defined as a thick intrusive sheet. A stock is a large irregular intrusive other than a laccolith, with more or less definite boundaries. A bathylith has no definite boundaries, but changes gradually into gneisses, etc.

Stocks.

Assuming the existence of a spherical intrusion, we might at once conclude that the three principal stresses were equal. This condition of stress comes about by the intrusive stress-modifications which we have previously mentioned, but the spherical form is superfluous, since we have already shown that an intrusion becomes irregular when the three principal stresses tend towards equality; *i. e.*, equi-pressure country gives rise to irregular intrusions.

In one illustration Geikie shows how a succession of eruptions has caused a complex structure of intersecting dikes, lying at several different angles, presumably the courses of previously-formed fissures, master-joints and cleavage.

Laccoliths (Figs. 7 and 16).

Geikie says of these bosses:

"Mr. G. K. Gilbert has described, under the name of "laccolite," a structure in the Henry Mountains in Southern Utah, which is probably not uncommon in denuded volcanic districts. Large bosses of trachytic lava have risen from beneath, but instead of finding their way to the surface, have spread out laterally and pushed up the overlying strata into a dome-shaped elevation (Fig. 16). Here and there, smaller sheets proceeding from the main masses have been forced between the beds, or veins have been injected into fissures, and the overlying and contiguous strata have been considerably metamorphosed." (*Op. cit.*, p. 571.)

These particular laccoliths do not seem to have dissolved or eroded the surrounding rocks to the usual extent. From the fact that this is a trachyte we may conclude that it

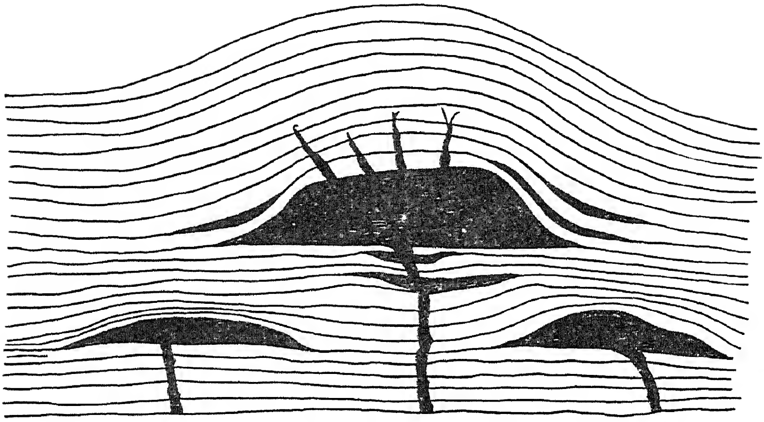


FIG. 16.—IDEAL SECTION OF LACCOLITHS.
(After Gilbert.)

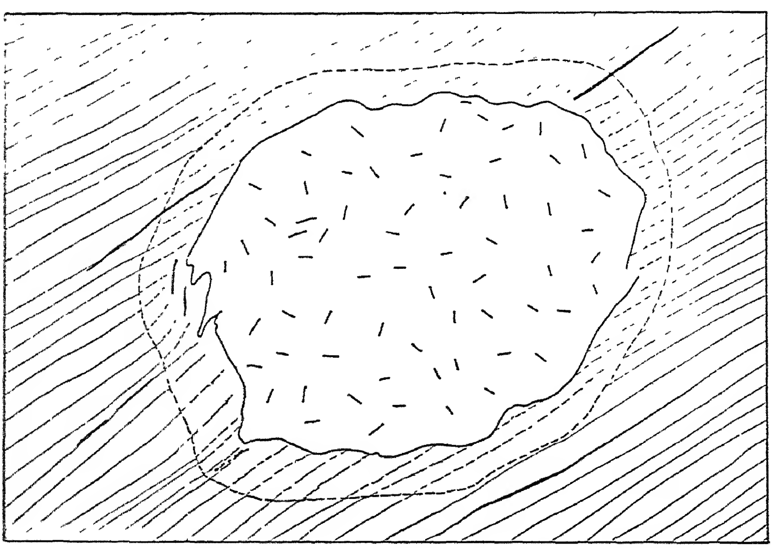


FIG. 17.—PLAN OF VERTICAL BOSS, CAIRNSMORE OF FLEET, SCOTLAND.
(After Geikie.)

must have cooled more quickly than if it had been a rock of coarser texture. Whatever may be the cause, we have admirable illustration of the origin of the usual laccolith. The dome-shaped lenticular form is, however, probably due to the cool, viscous character of the acid magma, and the corresponding laccolith of limpid base magma would be like the following description by Geikie:

"In Wales, as shown by the maps and sections of the Geological Survey, the Lower Silurian formations are pierced by huge bosses of different crystalline rocks, mostly included under the old term 'greenstone,' which, after running for some way with the strike of the strata, turn round and break across it, or branch and traverse a considerable thickness of stratified rock." (*Op. cit.*, p. 571.)

There seems to be little room to doubt that laccoliths are usually of secondary origin. Geikie makes a considerable argument in this direction:

"The remarks above made regarding the connection of intrusive bosses with volcanic action may be repeated with even greater definiteness here. Intrusive sheets abound in old volcanic districts, intimately associated with dykes and surface-outflows, thus bringing before our eyes traces of the underground mechanism of volcanoes. They frequently occur among the rocks that lie beneath a mass of ejected lavas and tuffs, or traverse the lower, sometimes even the upper parts of the volcanic mass. They then appear to mark some of the later stages of eruption when the orifices of discharge had become choked up and the subterranean energy only sufficed to inject the magma between the bedding of the rocks below ground but not to impel it to the surface." (*Op. cit.*, p. 576.)

Suggested Classification of Bosses.

The term "stock" was first used to designate large masses of mineralized matter, and I would suggest that it be confined to such use. Bosses could be better divided into vertical bosses, laccoliths, and skew bosses, so as to parallel the division of the smaller forms of intrusion, from which there is a gradual transition. Bathyliths may be called primary bosses.

Skew Bosses (Fig. 5).

Although principal stresses are so persistently vertical and horizontal, it may easily happen that, when intrusive stresses have made them all practically equal, any slight irregularities of the formation are liable to leave their impress on the shape of the intrusion. If the lines of weakness of the formation are inclined we shall have an inclined intrusion; if they are

irregular or absent the intrusion will be irregular. In either case we call the resultant intrusion a skew intrusion. Such a skew intrusion is enlarged to a skew boss by an upward lift of the overlying rocks, which movement does not of necessity have any horizontal component.

Vertical Bosses (Fig. 17).

This upward movement is restrained only by gravity. Vertical bosses cannot be subject to such free displacement of either of the walls. They are therefore, of necessity, necks and dikes enlarged only or principally by the melting or erosion of the walls by the magma which has ascended between them. As might be supposed, there is a gradual transition from ordinary necks and dikes into these vertical bosses, so that it is hard to say to which division some of them belong. It is also hard to say whether any large vertical boss originated from a neck or a dike; so that we can usually only call them vertical bosses.

Whatever was the original form of intrusion, we know that the surrounding rocks afterwards took on intrusive stresses. In this form they were not subject to any fissuring, so that the neck or dike once formed could not be caved in, no matter how deep it was. Any magma stresses in excess of the intrusive stress of the original neck or dike would cause new oblique and irregular dikes, sheets, and other small intrusions in the side of the original neck or dike. These would so weaken the sides as to cause them to be easily eroded, leaving a fresh surface for the abnormal pressure of the magma to work upon. A vertical boss is thus formed (Fig. 17). The great difference between Figs. 16 and 17 is to be noted. In the laccolite the overlying formation has been raised and the magma injected, while in the vertical boss the formation has been bodily excavated and taken away. The material eroded from the average laccolite boss might tend to give it somewhat the same characteristics as the vertical boss, but as a matter of fact geologists have noted the erosion most particularly in the latter form. Geikie says, for example:

"The manner in which some bosses of granite penetrate the rocks among which they occur strongly recalls the structure of volcanic necks or pipes. The granite is found as a circular or elliptical mass which seems to descend vertically through

the surrounding rocks without seriously disturbing them, as if a tube-shaped opening had been blown out of the crust of the earth, up which the granite had risen. Several of the granite masses of the south of Scotland, above referred to, exhibit this character very strikingly." (*Op cit*, p. 569.)

Intermediate Forms.

No absolutely distinct line can be drawn between original and secondary intrusions. Many may have been formed under stress-conditions which are intermediate between the kinds defined as primary and secondary. If we should try to make a determination by means of the history of the intrusion and the intruded formation, we would have too many indeterminate cases. We have therefore to resort to combinations of primary and secondary terms. We may thus make use of such expressions as "an irregular plane dike" or "an irregular vertically-corrugated dike."

Sub-Secondary Intrusions.

Large secondary intrusions often further modify the rock-stresses in the surrounding region. This is proved by the fact that smaller intrusions spring from them which are of a different kind from the secondary intrusion. The term "vein," generally used in this connection, corresponds exactly with the skew intrusion defined in this paper. Pegmatites and other residual products of solidifying magmas are sub-secondary intrusions. They have evidently been formed from limpid fluids, probably aqueous solutions, as they can run into small cracks in the formation. Geikie (p. 580) says of the pegmatites:

"Much discussion has arisen as to the origin of such veins. They evidently cut the ordinary granite and in so far may be regarded as intrusive veins."

Regular Intrusive Sheets.

In general, dikes will occur in regions of normal and side-thrust fissuring. Regular intrusive sheets, having the straight line of dikes, will, however, only occur in regions of over-thrust fissuring, and their rarity is comparable with that of true reverse faults. For these intrusions the stresses become very great, as appears from the stress-ellipsoids of Figs. 8, 9, and 10, it being always remembered that the vertical stresses are strictly proportional to the depth. With the accompanying

conditions of pressure, heat, and moisture, the processes of metasomatism, buckling, and crushing proceed to such an extent that the high ratios of greatest to least stress, necessary for fissuring, are not attained. Through the absence of these high ratios we also have an absence of the intrusions represented by Figs. 8, 9, and 10, and in their place we have those represented by the ordinary irregular laccolite (Fig. 7). It may be noted that the majority of steeply-inclined faults, which in a vertical cross-section appear to be reverse, are in reality side-thrust fissures.

Bathyliths (Primary Bosses).

Primary bosses arise through some form of primary stress which tends to prevent the magma from rising to the surface. Such forms are shown in Figs. 7, 8, 9, and 10, which all represent horizontal intrusions. But, as previously stated, of these 8, 9, and 10 are probably rare, the actual conditions of stress being usually more nearly represented by Fig. 7, or perhaps something between Fig. 7 and one of Figs. 8, 9, and 10.

It seems probable that these bathyliths continue to be expanded by the magma from some not very distant source, until they form enormous bosses, and possibly it often happens that no part of the magma ever comes to the surface.

Bathyliths are essentially deep-seated, occurring in folded and foliated rocks which at the time of intrusion possibly had a temperature almost that of the magma. Geikie gives several instances of primary bosses in gneisses, schists, etc.:

> "Granite frequently occurs in the central parts of mountain chains; sometimes it forms there a kind of core to the various gneisses, schists, and other crystalline rocks. . . . Sometimes it even overlies schistose and other rocks, as in the Piz de Graves in the upper Engadine, where a wall-like mass of granite, with syenite, diorite, and altered rocks, may be seen resting upon schists. In the Alps and other mountain ranges, it is found likewise in large bed-like masses which run in the same general direction as the rocks with which they are associated." (*Op. cit.*, p. 566.)

Stress-Regions.

From the laws of fissures, we find that the ratios of horizontal to vertical stress are least near the surface, and in general increase with depth. Therefore, if we leave out of consideration those cases in which the rock-stresses are modified by intrusion we get a mean order of depth for the occur-

rence of intrusions, as follows: Vertical necks (Fig. 1), vertically-corrugated dikes (Fig. 3), plane dikes (Fig. 2), horizontally-corrugated dikes (Fig. 4), irregular dikes (Fig. 6), skew intrusions (Fig. 5), and intrusive sheets (Fig. 7).

Horizontal Extent of Regions.

The regularity of breadth of plane dikes indicates the presence of remarkably constant horizontal stresses in the rock-mass at the depth of formation of most dikes. The great length of some dikes and the extent of some dike-systems show also that great areas of country are covered by regions of uniform stress-conditions. Relative to this Geikie says:

"Thus, in the south and west of Scotland, a remarkable series of basalt and andesite dykes can be traced across all the geological formations of that region, including the older Tertiary basalt-plateau. They run parallel to each other in a general north-west and south-east direction for distances of twenty and thirty miles, increasing in numbers towards the north-west, and they have been assigned to the great volcanic activity of Tertiary time. A dyke of the same series crosses the north of England, from near the coast of Yorkshire for about 100 miles inland." (*Op. cit.*, p. 583.)

General Classification.

A purely theoretical classification of intrusions might be framed on a plan similar to that proposed for fissures in my paper on that subject, as follows:

Class.	Sub-Class.	Variety.
Vertical.	Neck.	Regular.
Skew.	Plane.	Irregular.
Horizontal.	Vertical Corrugated.	Boss.
	Horizontal Corrugated.	

In this classification an intrusion of any class may fall into any sub-class, and similarly an intrusion of any sub-class of a class may be of any variety, so that there are 36 separate divisions into which an intrusion may fall without specific measurements being made or cognizance taken of the character of the filling matter.

Such a classification is entirely theoretical. On the one hand, it comprises forms of intrusion which rarely or never occur, such as plane skew and plane horizontal intrusions, and, on the other hand, it makes some distinctions which are too fine to be recognized in nature. Such, for example, as

the irregular corrugated skew intrusion, or the plane boss. Neither does this classification take cognizance of planes of weakness in the formations which give rise, with irregular forms, to the bedded intrusions, fissure-intrusions, cleavage-intrusions, and ramifying intrusions.

Partial Classification.

Out of the thirty-six theoretical kinds, we may therefore take the following eleven as natural and distinguishable:

Common Name.	Theoretical Classification.	Figs.
Neck,	Regular, neck, vertical,	1, 11, 12, 13.
Plane dike,	Regular, plane, vertical,	2.
Vertically-corrugated dike,	Regular v. corr., vertical,	3.
Horizontally-corrugated dike,	Regular h. corr., vertical,	4.
Skew intrusion,	Irregular, skew,	5, 15.
Irregular dike,	Irregular, vertical,	6.
Sill,	Irregular, horizontal,	7, 14.
Laccolith,	Boss, horizontal,	16, 14, 7.
Skew boss, } (stock),	Boss, skew,	5, 15.
Vertical boss, } (stock),	Boss, vertical,	17.
Primary boss (bathylith),	Boss,	7, 8, 9, 10.

Crushing-Machines for Cyanide-Plants.

BY MARK R. LAMB, MILWAUKEE, WIS.

(Canal Zone Meeting, November, 1910.)

THE recent growth of a sentiment among cyanide-plant designers against the use of gravity-stamps for the crushing preliminary to cyanidation may be said to date from the almost simultaneous perfection of the ribbed tube-mill liner and of the tall, air-agitation tank.

The first step in the resulting change of practice was the universal adoption of the tube-mill for fine-grinding and the use of coarse screening on the battery. This was the beginning of the surrender of supremacy by the gravity-stamp; since merely the single fact that it was a more economical machine for crushing fine had retained it in favor as compared with ball-mills, rolls, or steam-stamps. The surrender, however, is not yet complete.

It is the universal practice to use rolls where a relatively small tonnage is to be crushed coarse, as, for example, when coarse concentration is included in the plan of treatment. Power, attendance, repairs, and first-cost are smaller for the rolls, under such circumstances, than for stamps; and it is becoming daily more apparent that, with no concentration to provide for, the use of rolls and tube-mills, without the complication of trommels and elevators required by concentration, is rapidly gaining favor in the eyes of the leading designers.

The next step was the use of the rapid-running Chilean mill between the gravity-stamps and the tube-mill, which again narrowed the field of operation of the gravity-stamp while radically widening its screen-aperture. Such a combination of stamps and Chilean mill offers advantages when fine concentration is necessary, since the reduction in the Chilean mill can be adjusted easily and governed by varying the screen-mesh. But even concentration is now looked at askance. One of the leaders of this metallurgy has abandoned the separation and separate treatment of concentrates in the gold-mill which he is operating, and uses the concentrators merely as separators of a product requiring finer grinding than the average ore. The concentrate thus caught is continuously returned to the tube-mill, only that part of it escaping to the cyanide-plant which is too fine to be caught again on the tables and which is therefore in a proper condition for cyanide-treatment by air-agitation. Another consulting metallurgist has advised his clients to omit concentration in treating a silver-ore which is quite similar to that of Pachuca. At this point the old-style Chilean roller-mill has begun to come into its own, as, although expensive in first-cost, the power-, labor-, and repair-costs are very low.

The next step in point of time and the most radical in apparent daring was the abandonment of amalgamation of every product except the concentrates by the Goldfield Consolidated Co. in treating a high-grade gold-ore. Undoubtedly there were numerous reasons for this step; but not the least of these was the loss of gold through the operations of illicit gold buyers. The fact that the ore is so rich and the concentrate so very high in grade makes the change unusually noteworthy. Whether the above-described scheme of sliming the concentrate and treating it with the balance of the pulp

should be recommended in the Goldfield case, can only be decided by careful experiment. It is only fair to remember that last year the Meyer & Charlton experimented with a view to omit all amalgamation, both on mill-plates and on shaking-tables, recovering all the gold in the cyanide-works. This was reported a failure, but details are lacking. In any case, the Goldfield plant at present uses the stamps merely for coarse crushing, a work which probably could be done more cheaply with rolls, if the plant were re-designed; and the necessary machinery would cost less. Even steam-stamps would be cheaper than gravity-stamps, but the fuel-cost would have to be compared with that of local electric power.

In considering the steam-stamp, it should be borne in mind that there is no practical crusher of this type on the market with a capacity between the small Tremain, with its 20 or 30 tons daily to 8-mesh, and the stamp of the Lake Superior district, which crushes from 500 to 800 tons. This significant lack has dawned on several manufacturers at the same time: and the mine-owner with reasonable fuel-cost no doubt soon will have offered to him a simple steam-stamp of from 100 to 200 tons capacity, the cost of which will leave him little reason to prefer the gravity-stamp. The literature concerning the steam-stamp is strangely meager. There are scores of these monster machines in the copper country, and no secret is made of the operating- and repair-costs. The differences between the products of the various manufacturers, as well as between the simple and the compound stamp, are thoroughly well known, and the opinions of those locally in a position to know are in singular accord, particularly as regards the relative efficiency of the compound and the simple stamp. While, theoretically, the former has the advantage to the extent of 27 per cent., the operators will concede a bare 17 per cent., while with one voice they urge, as more than offsetting the saving of fuel, the disadvantages of the more complicated machine—difficulties of repairs and greater loss of time in tearing down to replace a broken cylinder-head or piston-rod. Moreover, while one man can attend to the operation of a simple stamp, three are required on some of the compound stamps. On account of mass-copper, a "human" feeder is required to be in constant attendance on these large units; but an automatic feeder will

be provided with the smaller units designed for crushing gold- and silver-ores.

For soft, or even for hard but friable ore, the cyanider should be encouraged by the success of the pebble-mill, taken from the cement industry, to adopt one more machine from the same source—namely, the ball-tube-mill. The disadvantages of the ordinary ball-mill which has been tried here and there on ore—its expensive liners and screens when used for fine crushing—are entirely avoided in the ball-tube-mill. What remains is merely a shell on trunnions, with simple steel plate or ribbed lining and a load of forged-steel balls. It may be of interest to state here that a 7- by 7-ft. ball-tube-mill weighs 54,000 lb. without its 12-ton charge of steel balls. The mill with balls sells for about $4,300, requires 140 h-p., and crushes 260 tons of dry clinker in 24 hr. from 1 in. to 20-mesh or a proportionately larger quantity to a coarser mesh. Clinker, as regards hardness, may be considered a rather soft ore, though in thinking of the work done in the mill, any piece of ore is soft under the blow of a 6-in. steel ball weighing 30 lb. and falling 5 or 6 ft. In a delightfully-frank contribution to the *Mining Journal* of Sept. 4, 1909, M. W. Von Bernewitz gives interesting data on dry-grinding in ball-mills of the old type, provided with screens. He breaks 120 tons in 6 hr. in a No. 5 Gates, and puts the 3-in. product through three No. 5 ball-mills to 27-mesh. This dry-grinding costs 42 cents per ton, each mill requiring 16 to 20 h-p. The wear is 3.5 oz. of steel per ton of ore ground.

It will be seen from all the foregoing that there is a choice among three machines, to replace gravity-stamps advantageously in cases where neither concentration nor amalgamation is necessary. The columns of the technical press show that the simple nature permitting such treatment is attributed to an increasing variety of ores. In fact, except for the occasional manganese-silver ore, it might be said that cyaniding is a purely mechanical process, requiring a chemist merely as a form of insurance. A simple crushing- and grinding-plant would consist of breaker, ball-tube-mill, classifier, and pebble-tube-mill, in which attendance and skill-requirements would be reduced to a minimum. If rolls are preferred to the ball-tube-mill, their substitution does not affect the other ma-

chines. If fuel is advantageously cheap, the plant will consist of crusher, steam-stamp, classifier, and tube-mills. For small plants or with unskilled labor the slow Chilean mill has advantages. In either case, there is no work for the gravity-stamp which cannot be done better by either of the other coarse crushers.

We may rest our case against the gravity-stamp in general terms as follows: Large tonnages of hard ore are crushed advantageously to $\frac{3}{16}$ in., or coarser, with steam-stamps. Steam-consumption, labor, repairs, and first-cost are smaller than the corresponding items for gravity-stamps. Such steam-stamps are for capacities exceeding 500 tons daily. Small tonnages of hard rock are crushed advantageously with rolls, which are admittedly cheaper than gravity-stamps in first cost, repairs, labor, and power, if they are not expected to crush too fine, and are not complicated with screens and elevators—as they need not be for feeding tube-mills. Large ball-tube-mills are available for reducing 3-in. ore to a size suitable for feeding pebble-tube-mills in one operation with no screening. Numbers of these mills are in use for crushing hard, glassy clinker. The objection may be urged that tough quartz is very different from clinker. So it is; but the same objection was urged only a few years ago against the use of the pebble-tube-mill and was found to be groundless, except that the capacity of that mill is smaller with quartz than with clinker. Finally, small steam-stamps will undoubtedly be at the disposal of the metallurgist within a short period. This machine needs no explanation or justification, since it will merely fill the gap in sizes of a type of mechanism, the merits of which have been proved in practice.

The entire mechanics of the cyanide process have changed radically within a short period. While we in America have wondered and worried because South Africans were building heavier stamps than we could be persuaded to use, perhaps, after all, we have been luckily slow.

Recent Developments in the Undercutting of Coal by Machinery.*

BY EDWARD W. PARKER, WASHINGTON, D. C.

(Canal Zone Meeting, November, 1910.)

I. INTRODUCTION.

AT the Seventy-sixth meeting of the Institute, held in New York, N. Y., February, 1899, I presented a paper on this subject entitled, Coal-Cutting Machinery,[1] which has become somewhat out of date, in view of the decade's development in this feature of the coal-mining industry. It should be explained that the present paper is limited to the use of mining-machines in bituminous mines. Practically all of the anthracite mined is "shot from the solid," no undercutting being done. Anthracite-mining is therefore not considered in the statistics or other references.

The statistical record as compiled by the U. S. Geological Survey shows that in 1898, the year preceding the presentation of my former paper, there were 32,413,144 short tons of coal undercut by the use of machines. In 1907 the machine-mined coal-product amounted to 138,547,823 short tons. In sympathy with the general decrease in coal-production during 1908, the output of machine-mined coal decreased to 123,183,334 short tons, but the proportion of machine-mined coal to the total output, nevertheless, showed an increase, from 35.1 per cent. in 1907 to 37 per cent. in 1908. In 1898 only 19.5 per cent. of the total product was undercut by the use of machines. The number of undercutting-machines in use in the bituminous mines of the United States has increased from 2,622 in 1898 to 11,144 in 1907, and to 11,569 in 1908. The total production of bituminous coal in the United States in 1908 was almost exactly double that of 1898, while the machine-

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[1] *Trans.*, xxix., 405 to 459 (1899).

mined portion of the product of 1908 was nearly four times that of 1898. These figures give an idea of the growth in the mechanical production of bituminous coal. It is unfortunate that we have no exact information in regard to the quantity of pick-mined coal produced. It is not sufficient to say that the difference between the total production and that reported as mined by machines represents the pick-mined tonnage, since there would be included in this difference a much too large quantity of coal mined principally by the use of powder. And right here is one of the most serious problems with which the bituminous-coal operators of the United States are confronted at the present time. The undercutting of coal by hand with a pick is an exacting kind of labor and the miner may not be blamed for a disinclination to lie for 5 or 6 hr. on his side making a cut of from 4 to 5 ft. deep in the coal. When, however, conditions are such that the mechanical cutting of the coal is impossible, no other considerations should permit the miner to neglect his duty.

There has been in the bituminous regions a growing tendency to shoot the coal "from the solid," which consists of "making the powder do the work," a practice which cannot be too vigorously condemned. It is bad for many reasons, the least of which is the profligate waste arising from the excess of slack or fine coal produced, a product which is frequently unsalable, or, unless suitable for the manufacture of coke, must be sold at less than the cost of production. Moreover, it is pre-eminently anti-conservational. Far worse than this is the increase of the natural hazards to which the mining of coal is subject. To all familiar with bituminous-coal mining, it is well known that when "the powder does the work" the tendency to "windy" or "blown-out" shots is markedly increased and that these are prolific causes of the dust-explosions which so often fill with horror the readers of the daily press. Few of the reading public realize, however, that a far larger aggregate of deaths in the coal-mines is due to the falls of slate and coal, and that these are indirectly due to the same originating cause. In 1907, the darkest year in the history of coal-mining in the United States, so far as the number of lives sacrificed goes, there were 947 men killed in gas- and dust-explosions, with 343 injured, while the falls of coal and

roof killed 1,122 and injured 2,141. In 1908 explosions killed 396 and injured 326, and falls of coal and roof killed 1,080 and injured 2,591. The figures are significant, but it is hard to make the miner see that the weakening of the roof and the fracture of the ribs by the excessive use of powder in his work are responsible for the falls which make the long lists of casualties every year. Mining engineers know it, but it is a hard thing to prove it to the miner, or if he, too, knows it, he still assumes the risk. The public attention is not called to it in the same way that the horrors of explosions are heralded, for these accidents from falls, which usually occur singly, do not figure in the press-dispatches, and are not known outside of the community in which they occur.

A contributing factor, which has for one of its results the continuation and the extension of this reprehensible practice of shooting from the solid, is the custom in some coal-mining districts of paying for the mining of the coal on the mine-run basis. Where such custom obtains there is no incentive to the miner to produce a merchantable grade of coal. He is paid as much for slack as for good coal, and when "the powder does the work," what's the odds? In one State, Oklahoma, this pernicious influence has been so highly developed that by act of legislature the operators are compelled to pay for the mining of coal on the mine-run basis. Can any one imagine that the legislators of that State realize to what an extent they have made themselves responsible for the accidents that are sure to occur? And I am credibly informed that there is strong likelihood of no less a bituminous-coal-producing State than Pennsylvania writing a similar law upon its statute-books. Of what use, may we well ask, to establish mine-rescue stations, unless to recover the dead bodies, if law-makers show such ignorance of the conditions which make towards the saving of life and limb, the protection of property, and the conservation of the coal. The coal-miner is one man who has to be protected not only against himself, but more particularly from the harmful activities of his would-be friends.

Let me give one illustration of the evil effects on the coal itself of paying the miner on the basis of run-of-mine coal. One of the large operators in the interior region has furnished me a statement that shows some striking comparisons. The

company operates in several States, and its production each year exceeds 1,000,000 tons. The mine-run basis was put into effect on May 30, 1900. The records kept by the company show that from 1894 to 1900 the proportion of slack was 27.8 per cent. of the total output; from 1901 to 1905, with mining on the mine-run basis, the percentage of slack was 36, varying from 32 per cent. in 1901 to 38.15 per cent. in 1904. Since 1905 the company has made an agreement with the railroads by which the latter accept, as lump-coal, the mine-run product with 25 per cent. of slack screened out. This arrangement has apparently reduced the percentage of slack, but still, from June 1, 1905, to May 31, 1909, the screenings have equaled 32.36 per cent. of the total production.

It is a self-evident proposition that the payment for this waste must eventually fall as well upon the miner as upon the operator and the consumer.

As the sewing-machine has supplanted "the bright little needle, the swift-flying needle, the needle directed by beauty and art," and the machine reaper has taken the place of the hand-swung cradle on the farm, so has the coal-mining machine ameliorated to a marked degree the labor-conditions in the coal-mines. A production by the use of undercutting-machinery of 138,500,000 tons, or 37 per cent. of the total output of bituminous coal, is significant. A portion, and probably a large portion, of this large quantity, had it not been undercut by machines, would certainly have been shot from the solid; and every ton of gain in machine-mined coal is that much more for better coal and greater safety for the mines and the mine-workers, notwithstanding the fact that the operation of the machines themselves is occasionally responsible for some of the casualties. The mine-inspectors' reports contain statements of the men killed and injured by the mining-machines, but these do not and cannot subject to risk the hundreds of other employees, as is done every time a shot is fired from the solid.

While the anthracite-mines of Pennsylvania are not liable to the same danger of dust-explosions as are the bituminous mines, the excessive use of powder by the miners, about 30 years ago, became a serious question, because of its effect upon the coal and upon the roof and pillars; and in order to minimize the evil, the operators, by agreement with the miners, increased

the price of powder to a figure which made it an incentive for the miner to exercise economy in its use. The increase in the price of powder was compensated to the miner by an increase in wage. Nearly 25 years later, when the manner in which the adjustment had been made had passed out of memory, the "extortion" practiced by the operators became the subject of somewhat acrimonious discussion, and in order to settle the difficulty the practice was abandoned, whether wisely or unwisely I am not prepared to say.

In the mining of bituminous coal, proper regard for safety, and the securing of the maximum quantity of marketable coal, require that the minimum amount of powder necessary to bring down the coal should be used. This establishes as a pre-requisite that the coal should be properly undercut and sheared. There are, of course, some mines in which the physical conditions militate against the introduction of mining-machines, but these are comparatively rare.

A recent development in mechanical coal-cutting has been the design of a machine adapted to use in steeply-dipping beds as well as those in which the inclination is slight. This machine is described in detail later in this paper. But whatever the physical conditions, the coal should be undercut, and no excuse for failure so to do should be accepted. At the present time the only mines in which the installation of mining-machines is impracticable are those of comparatively small production, and of a capitalization and market-conditions which do not permit the necessary expense. These mines, however, are becoming fewer each year. The tendency in coal-mining, as in other lines of industry, is towards the large unit, for it is only in the operation of large units that effective economy in production and in the marketing of the product can be secured. The average production of the bituminous coal-mines in the United States in which undercutting-machines were employed, in 1908, was 94,177 short tons. Exclusive of the purely local banks, whose output in no case exceeded 1,000 short tons, the average production from mines in which machines were not used was 45,281 short tons, and including the output of local banks, this average is reduced to 29,442 short tons. It is claimed that many more machines would be used and that a much larger machine-mined output would be mined but for the differential in wages

made against the machine-mined coal. This, it is averred, takes from the operator the earning capacity of the capital invested in the installation, and limits the advantage secured by him to the larger proportion of lump-coal obtained by having his coal undercut.

The two general types of undercutting-machines most in use at the present time, as 10 years ago, are the pick-, or puncher-, and the chain-breast machine, although some long-wall machines are in use in the thinner beds, principally in the interior States. Long-wall mining is not practiced in the United States to the same extent as in Europe, nor to the extent it should be.

In 1908, of the 11,569 machines in use, 6,380 were of the pick, or puncher, type, 4,992 were of the chain-breast pattern, and 197 were long-wall.

II. MACHINES OF THE PICK, OR PUNCHER, TYPE.

In general principles this type of undercutting-machine has not changed in the past decade, but considerable development has been made in the application of this type of machine to steeply-dipping beds and the adaptation of electricity to its operation. These have been accomplished in connection with steeply-dipping beds by the construction of a machine that, attached to a post, may be used either for mining or for shearing, the machine rotating on the post, but otherwise operated in much the same manner as the air-drill. In this way we have the Whitcomb post machine, the Sullivan post puncher, and the Ingersoll radialaxe machine, with which, it is claimed, coal can be undercut or sheared, let the pitch be what it may, even to the vertical. The application of electricity has been accomplished by utilizing compressed air for striking the blow. Since these represent the latest development in the punching-machines they will be given in this paper somewhat more extended treatment than the older patterns.

The successful application of electricity to the puncher-machine is worthy of particular attention, for the reason that, since the advent of electricity in coal-mining, numerous attempts have been made to apply it to machines of the puncher type. Great difficulty, however, has always been encountered in producing a blow of sufficient strength to do the work and at the same time make the machine easy to handle in the hands

of the machine-runner. ' Attempts to produce a purely electrical machine of this kind have met with failure, as the solenoid machine, which represents the only purely electrical development, has not been found at all practical. Attempts to produce a blow by compressing a spring, by means of gearing, cams, or other means actuated by an electric motor, have also failed. The successful machines which are now being manufactured make use of compressed air and to that extent may be considered as not purely electrical. The machine, however, is self-contained; the electric motor and the air-compressing parts being combined in one machine, which, in appearance, closely resembles the compressed-air punchers. The pneumelectric represents the combined electric and air coal-puncher. Experiments have also been made with the Temple-Ingersoll electric-air combination in the operation of puncher-machines, but at the present time its application is confined entirely to rock-drills and similar devices. The principal advantages claimed for this type of machine over the ordinary or old-time puncher are that the expensive installation of conducting-pipes and hose, from the compressor on the surface to the machine at the face, is avoided and the power-consumption is very materially reduced.

The first coal-punching machine used in the United States was the invention of J. W. Harrison, of Adrian, Mich.,[2] who later patented certain improvements.[3] The machine, Fig. 1, was first presented to the operators by the George D. Whitcomb Co., of Chicago, in 1880. It was natural, of course, that there should be some prejudice to overcome. Other machines had been tried before, and as in all new departures, most of the productions had proved failures, and operators who had installed them had only the expense-account to show for the results. Opposition of the miners' unions to the supplanting of hand labor by mechanical methods had also to be overcome. It was not long, however, before the advantages of the machine- over the muscle-driven pick had been demonstrated, and after the first year or two of educational campaign the manufacture of coal-cutting machinery was a fixed and rapidly-growing industry.

[2] U. S. Patent No. 198,610, Dec. 25, 1877.

[3] U. S. Patent No. 219,090, Sept. 2, 1879.

Pick-, or puncher-, machines exclusively air-driven are now manufactured by the following concerns: George D. Whitcomb Co., Rochelle, Ill. (formerly at Chicago); Sullivan Machinery Co., Chicago, Ill. (works at Chicago and at Claremont, N. H.; this company also manufactures electric chain-breast and long-wall machines); Ingersoll-Rand Co., New York, N. Y. (works at Philipsburg, N. J.): and Herzler & Henninger Machine Works, Belleville, Ill. (Latterly this company has devoted its attention chiefly to the manufacture of cages, screens, and mine-cars, and has not been developing coal-mining machines.)

As is shown by Figs. 1, 2, and 3, there is little difference in the general style of the various makes of machine. Such differences as exist are chiefly in minor details, such as valve-action or method of air-control, and need not be discussed here.

The puncher-machines are operated on a running-board, 8 or 9 ft. long by 3 ft. wide, the back end of which is elevated from 4 to 20 in., in order to give an inclination towards the coal. The air is conducted from the supply-pipe to the machine by a length of hose, usually about 50 ft. The crew for one machine consists of two men, the machine-runner and a helper, it being the duty of the latter, after the machine is set up and ready to operate, to keep the cut clear of the cuttings. The first, or "sumping," cut is usually at the left side. As this is cut from the solid, and the operator has no open side to cut to, it takes somewhat longer than the subsequent cuts. The cut is V-shaped, being from 8 to 10 in. high in front and tapering to 2 in. in the rear. The advantage of having the cut higher in front than in the rear lies in the fact that the coal when shot down does not bind, but falls away easily from the face.

After the sumping-cut is made, the machine is run off the running-board, and the board is shifted into position for the second cut, and so on. Each cut will average from 4.5 to 5 ft., the depth varying with the thickness of the bed up to 5.5 or 6 ft. After a room or entry has been undercut entirely across the face, the crew of two loads the machine and board upon a truck, on which it is pushed to the next working-place. One man and his helper can undercut several rooms in a day, the number depending upon the hardness of the coal and the width of the rooms.

Machines of the puncher type may be used in mines in which, on account of weak roof, it is necessary to keep the props close up to the face. These machines also have the advantage of being able to cut around sulphur-balls or other hard substances in the coal. They can be mounted on high trucks and used for shearing, if desired. If electricity be not used there is no danger of dust- or gas-explosions resulting from an electric spark. It is claimed that a pick-machine or puncher will do the work of from 6 to 15 skilled miners working with hand-picks. They have been operated on beds having an inclination of 15° from the horizontal. A refinement in the manufacture of the machines is the standardization and interchangeability of the parts, so that supplies for repairs on all machines of the same make may be kept in stock, and do not have to be ordered from the factory. The details of the superiority of each make of machine over all others may be obtained upon application to the several manufacturers.

As previously stated, the most recent application of the principle of the puncher type of machine is to the mining of coal in steeply-inclined beds. It is in reality a combination of the air-driven rock-drill and the punching-machine, with this essential difference, that the rock-drill operates only in one direction at a time, driving a straight round hole, while the machine adapted to coal-mining rotates or pivots on its supporting-post, and cuts a groove under or into the coal, the farther limit of the groove being the arc described by the tool rotating on the post to which it is fixed. The machine can be changed quickly from a mining- to a shearing-machine by simply changing the adjustment on the supporting-post. In one respect it outclasses the regular, or old style, type of puncher. It can be adjusted on the post so as to cut out a clay band near the top, in the middle, or near the bottom of a coal-bed, whereas the older type running on low wheels is only adapted to the undercutting of the coal.

Three companies have placed this type of machine on the market. All of them are also manufacturers of puncher undercutting-machines. The Sullivan Machinery Co. manufactures the Sullivan post puncher; the George D. Whitcomb Co. makes the Whitcomb post machine; and the Ingersoll-Rand Co. produces the radialaxe coal-cutter. The following descrip-

tions and claims have been furnished by the manufacturers: The description of the Sullivan puncher is extracted largely from an article entitled, Working a Steep Coal Seam,[4] written by Austin Y. Hoy, of Spokane, Wash.

1. *The Sullivan Post Puncher.*—The particular machine described was installed in the Coal Creek mine of the Pacific Coast Coal Co. The coal bed is 4.5 ft. thick and pitches at an angle of 38°. The coal is non-coking, and the practice previously in vogue of "shooting from the solid" made a large quantity of unsalable coal.

The post puncher is built on the patents of the English Siskol coal-cutter, and resembles a rock-drill in the fact that it is mounted on a post or column, and uses length-bars, or steels of various lengths, in doing its work. It resembles the puncher- or pick-machine in its ability to mine across the face, or to shear a room from top to bottom. For this purpose the machine and shell are mounted on a gear-segment, controlled by a worm and crank. In mining, the operator swings the bit in an arc back and forth across the face. By loosening one nut, the segment may be set in a vertical plane, and a shearing-cut made in a similar manner. This feature is a great convenience when the coal will not shoot readily unless both mined and sheared. A valuable feature of the post puncher is the fact that the machine does not swing around the post, but around the center formed by the socket, in which it is clamped, about 12 in. from the post. It is, therefore, unnecessary to set the column exactly square with the intended mining, but the machine may be set in a few seconds to mine wherever desired, regardless of careless placing of the post. This is a patented feature, possessed by no other similar machine. Dismounting the machine when swinging from the mining- to the shearing-position is also unnecessary.

The cylinder and valve-motion embody other novelties. The front head is solid with the cylinder, thus eliminating the complication of side-rods. When the coal is missed, the piston will over-run the front port and stop, thus obviating damage to the front head. This over-running seldom occurs with a skilled operator, but the delay attendant on feeding the machine against the coal, so that it will start again, is nothing to the delay due to breakage that would occur eventually if the piston could strike the front head The cylinder is made of steel, to resist the heavy shocks and strains received from falling coal. Felt packing of a durable character and having an adjustable gland is employed. A combined shell-adjustment and stop, to prevent feeding too far, and a rotation without springs, provided with five round steel pins or pawls, are other interesting features of the post puncher design.

The exhaust-cap is so arranged as to throw the air in any desired direction to avoid raising dust. The extension-rods, or length-bars, range from 20 to 100 in. in length, and are tapering at both ends, one to fit the chuck and the other to receive the bit-holder. This bit-holder has tapered holes for from three to seven chisels or bit-points. A solid bit may be used instead, depending on the nature of the cutting.

The mining may be put in at any desired height in the seam, so as to mine in fire-clay, the coal itself, or in a dirt band, by raising or lowering the clamp on the post. . . . .

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[4] *Mine and Quarry*, published by the Sullivan Machinery Co.

In operation, the first cut is made from a post set, say, 7 ft. from the left rib and about 18 in. from the face. A mining 7 ft. in depth is put in, using an extension-bar 80 in. long. The chuck enters the cut, which accounts for the mining being deeper than the length of the extension. After the 80-in. extension has been swung, a 100-in. bar is used to square up the cut. One man operates the machine, swinging it by means of the worm-crank with one hand, and feeding the cylinder forward two or three turns with the other, at each end of the swing, as shown in Fig. 4.

Although in a steeply-dipping bed the cuttings fall out of the cut without scraping, due to the pitch, a helper is required to set up the machine, pick down coal, etc. While the operator is completing the left-rib cut, the helper sets up a second post about 10 ft. from the first one, or about 17 ft. from the left rib, and 18 in. from the face. Upon finishing the rib-cut, the machine and swinging-attachment are transferred to the second post with only a few minutes' delay. While the operator is making the second cut, the helper takes the first post down and resets it about 27 ft. from the left rib for the third cut, and so on, until the face is crossed. Only a half swing is used, so that the operator is protected by the un-mined coal, as well as by the machine itself, from the coal loosened from the face by the mining.

The machine and posts remain in the room at the face until the room is completed, for there is no shooting of the coal that can injure the machine, nor any loading (as in a flat seam) that the machine would interfere with. Hence there is no waste of time due to moving, except from post to post, and little heavy lifting. Two men can set up with ease, as the heaviest parts (the machine and shell) weigh only 225 pounds.

While the Sullivan post punchers are particularly adapted to heavily-pitching seams, they are also useful in flat seams, where it is desirable to mine in a dirt band above the floor or near the roof. In seams where the cuttings do not fall from the mining by gravity, the helper keeps the cut free by means of a long-handled flat shovel or a scraper.

In some fields these machines have been successfully used for driving entries, and for shearing they can hardly be excelled, especially when both mining- and shearing-cuts are required, owing to the quickness and ease with which the setting may be altered for either purpose.

It has been noted that the post puncher was used at Coal Creek with only a half swing, and that the posts were therefore set only 10 ft. apart. In stronger coal, or in a flatter seam, where the danger to the operator and machine from falling coal is less serious, the machine is used to cut on both sides of the post, from the same setting, having a maximum capacity of about 18 ft. of face from one set-up. To keep the ribs or side-walls of an entry or room straight, the bit is swung back and forth across the corner, gradually advancing it with the feed-screw until the angle is square. Or a hole may be drilled at the extreme end of the swing to the full length of the feed-screw, then the bit cranked back and the mining made up to this hole at each swing.

The post puncher may also be employed for wedging-down coal in mines where the use of explosives is prohibited. After undercutting and shearing, the machine drills a hole about 3 in. in diameter at the top of the seam. An ordinary bit may be used for this purpose. The hole is then scraped out and a compound wedge (plug and feathers) inserted. The machine may be swung around the post, if necessary, to give access to the hole. When the plug and feathers are placed the machine is swung back to position opposite the hole, and a special hammer inserted in the chuck. The air is gradually turned on, the hammer striking the wedge with increasing force until the coal breaks down.

2. *The Whitcomb Post Machine.*—The Whitcomb post machine was designed to meet certain conditions existing in some coal-mines to which the standard punching-machine is not adapted. Primarily it was designed for the purpose of cutting out dirt bands which, if shot down with the coal, become so mixed with it that unless afterwards separated by washing the quality of the whole output is injured. Washing adds to the expense, and as a usual thing a considerable quantity of coal goes off with the refuse. The dirt bands often come in the center or upper half of the coal-bed, where it is practically impossible to cut them out with the standard mining-machines. The machine is also designed for entry-driving, where more speed can be secured by putting in a vertical cut, or "shear,"

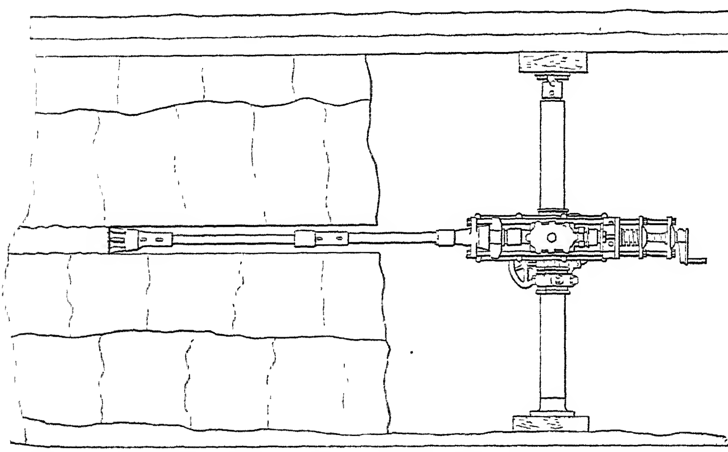

FIG. 5.—METHOD OF CUTTING OUT DIRT BAND IN MIDDLE OF VEIN, WHITCOMB POST MACHINE.

and shooting to the shearing. The machine is built on the general lines of a rock-drill, but is mounted so that the machine may be moved through a half-circle while in operation, and cuts a channel instead of a hole. The cut is from 3 to 6 in. wide and from 3 to 8 ft. deep, as the case requires.

For shearing, or putting in a vertical cut, a single-motion mounting is used. The machine is mounted on a heavy steel trunnion, or axle, on which it swivels. The motion is given by a worm which engages a worm-gear on the saddle of the machine. The worm is driven by two small miter-gears, in ratio of three to one, which makes it easy for the operator to move the machine. All parts of the mounting are made of

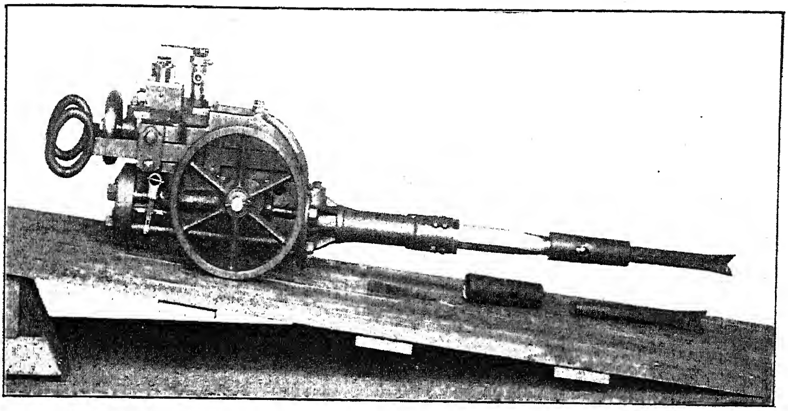

FIG. 1.—HARRISON MACHINE, 1883 TYPE.

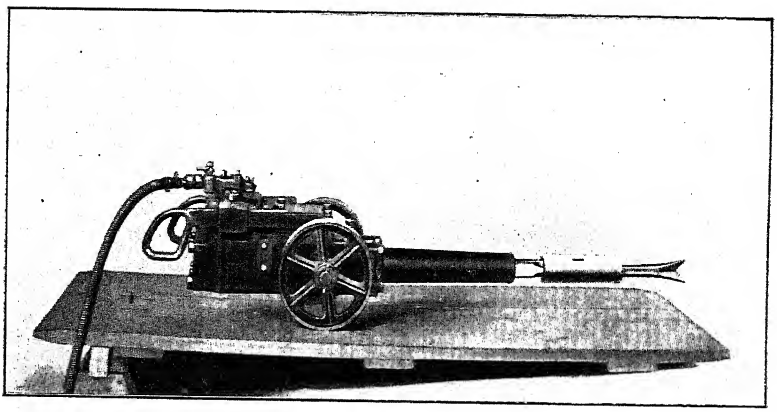

FIG. 2.—SULLIVAN PUNCHING-MACHINE.

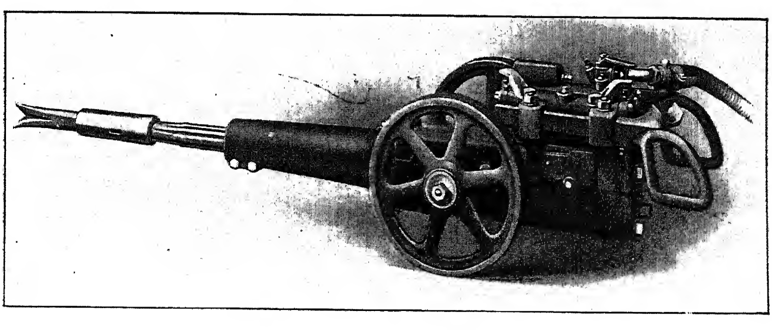

FIG. 3.—NEW INGERSOLL COAL-CUTTER.

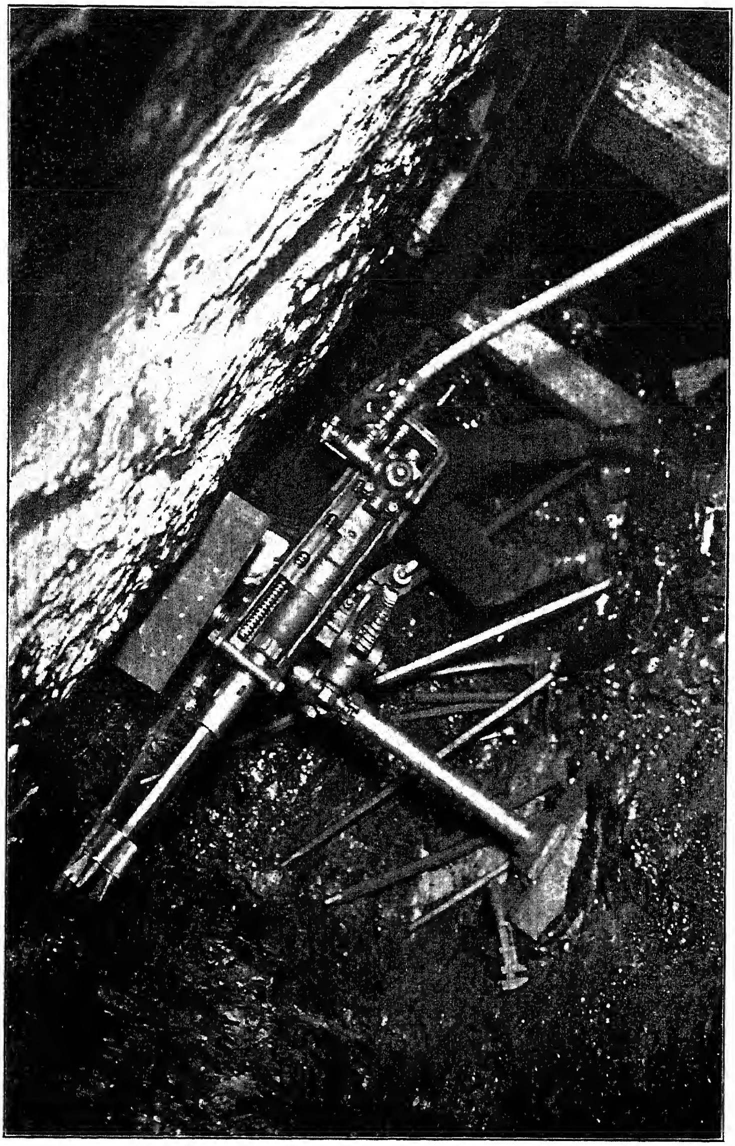

FIG. 4.—SULLIVAN POST PUNCHER BEGINNING CUT NEAR ROOF IN A 4-FT. BED, 38° PITCH.

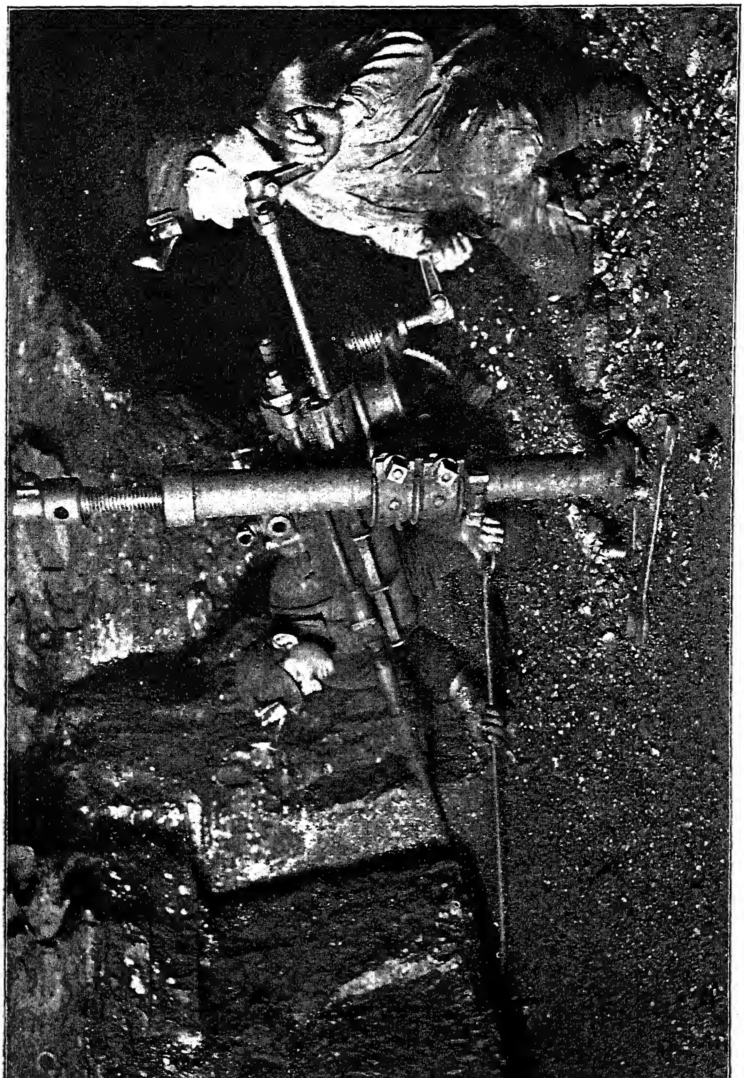

Fig. 6.—Ingersoll-Rand Radialaxe, Shearing.

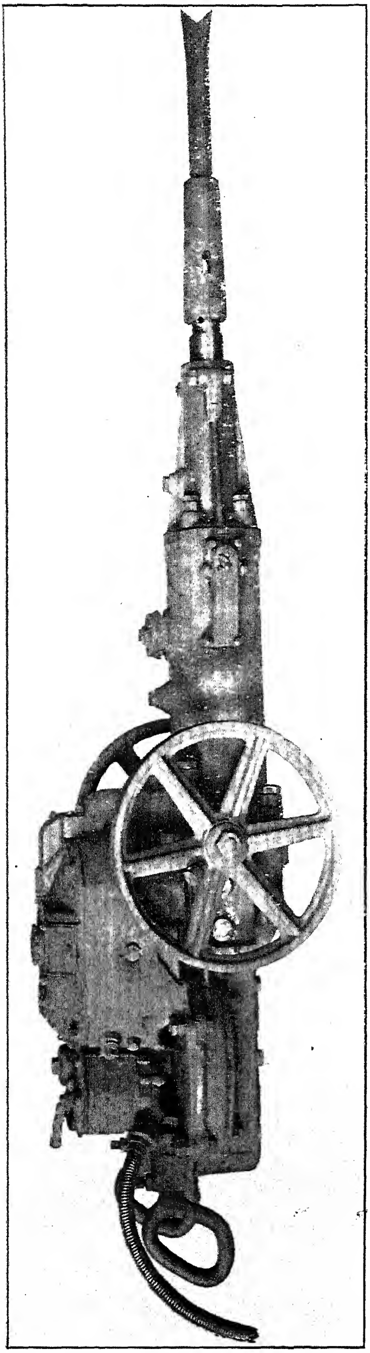

Fig. 7.—Pneumelectric Machine.

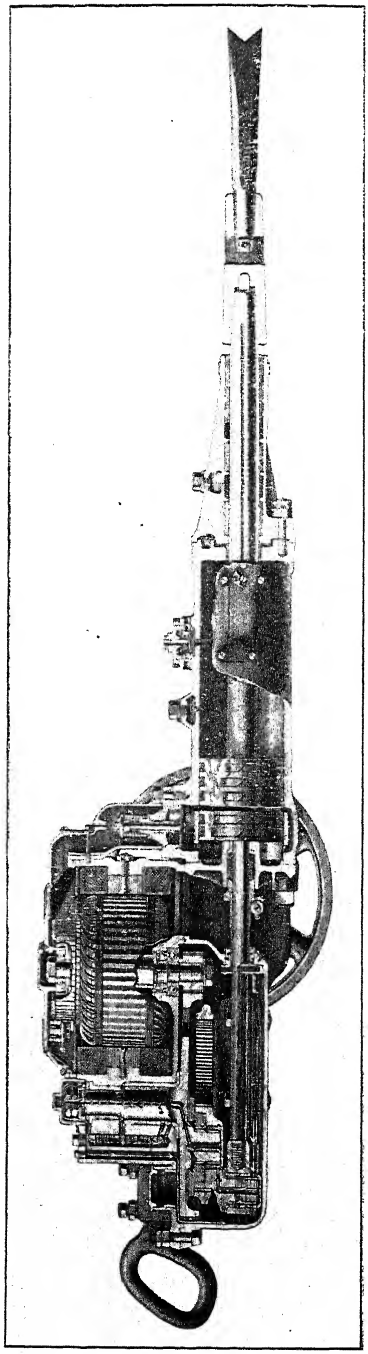

Fig. 8.—Pneumelectric Machine, Sectional View.

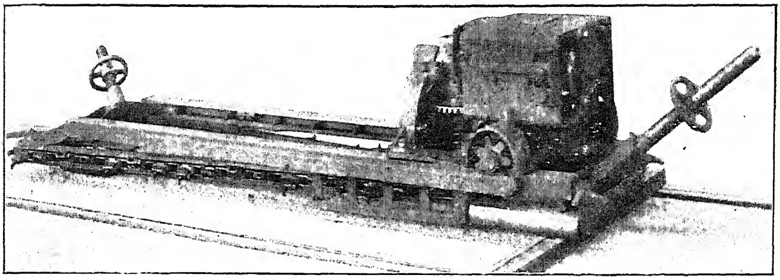

FIG. 9.—GOODMAN CHAIN-BREAST MACHINE.

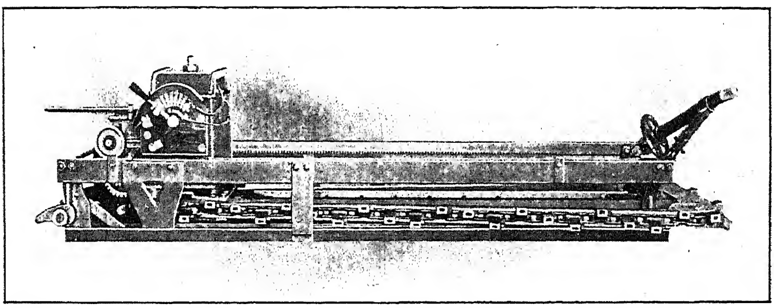

FIG. 10.—MORGAN-GARDNER CHAIN-BREAST MACHINE.

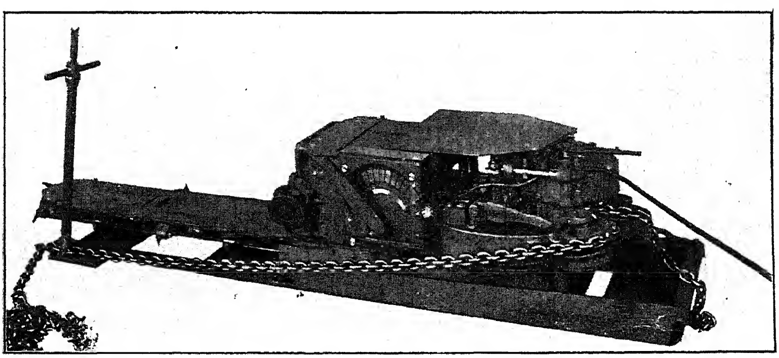

FIG. 11.—SULLIVAN CONTINUOUS CUTTER, IN STARTING FRAME, READY TO MAKE CUT.

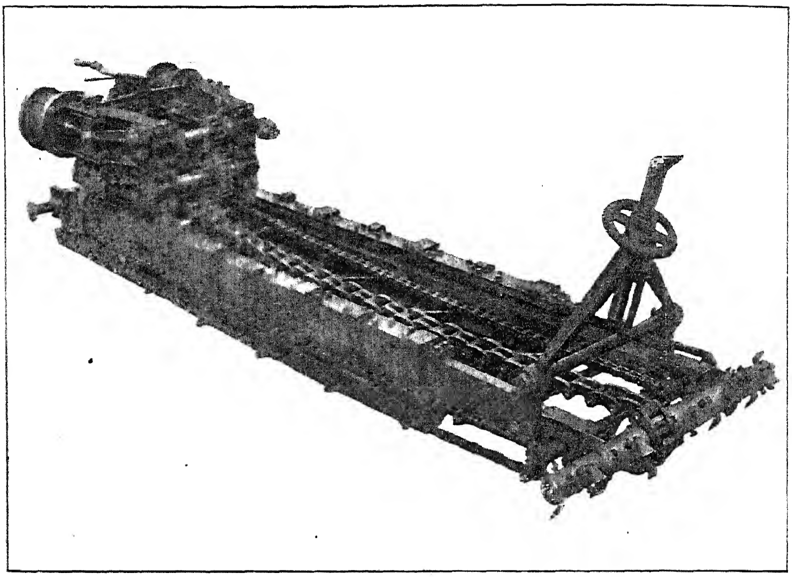

FIG. 12.—JEFFREY CUTTER-BAR MACHINE WITH HORIZONTAL ENGINE.

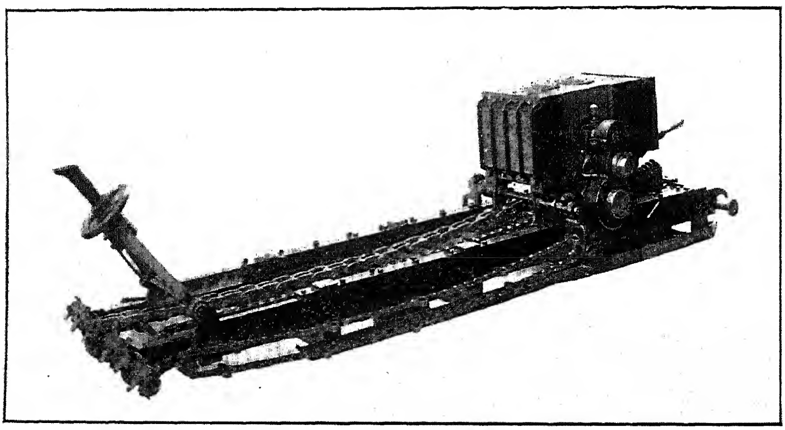

FIG. 13.—JEFFREY CUTTER-BAR MACHINE WITH ELECTRIC MOTOR.

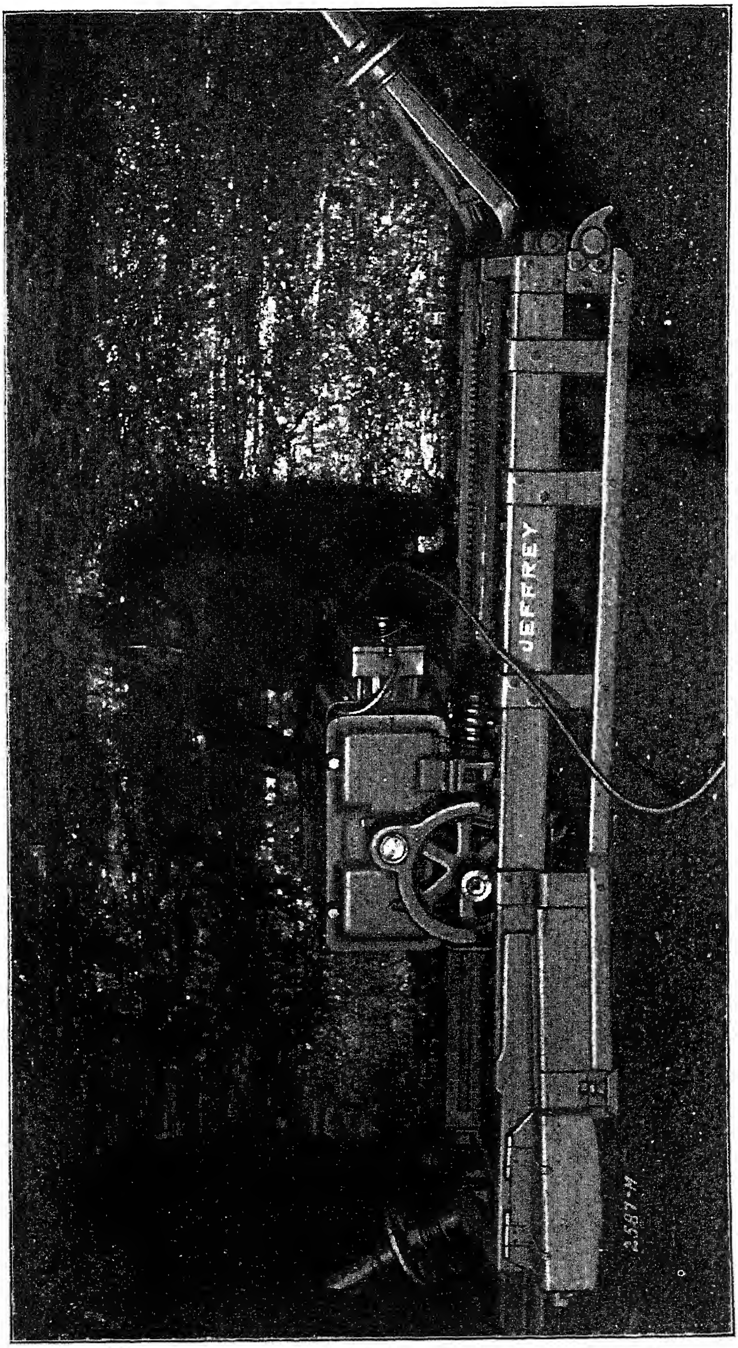

FIG. 14.—JEFFREY CHAIN-BREAST MACHINE, FLAME-TIGHT MOTOR.

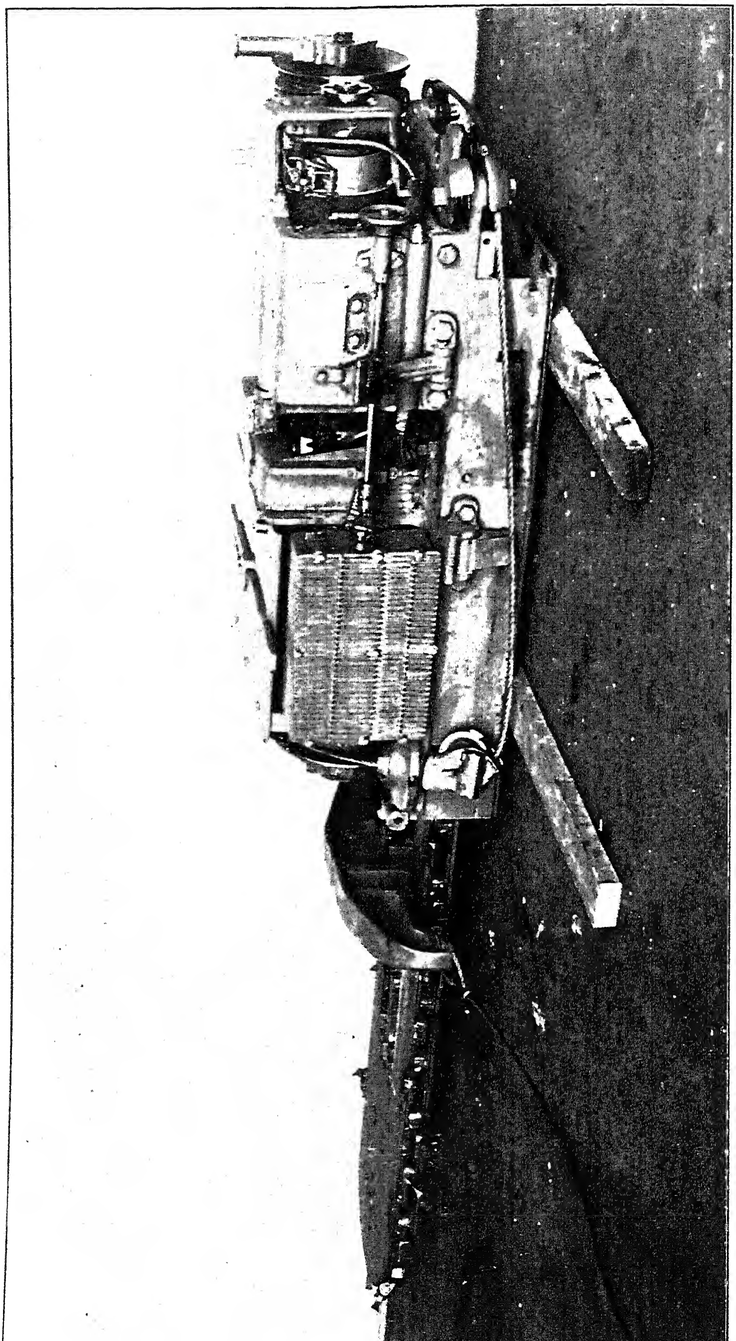

Fig. 15.—Jeffrey Short-Wall Machine.

steel, giving great strength and practically eliminating any danger from breakage. The columns are furnished with either single or double screws, as may be desired.

For cutting out a dirt band, either in or over the coal, and for making a shearing-cut without changing or remounting the machine, the double-motion mounting is used. This is the same as the single mounting, except that instead of being held to the column by a clamp it is mounted on a sleeve that encircles the column. The sleeve is provided with a worm-gear on the lower end that meshes with a worm held in a bracket clamped around the column. By means of the worm and worm-gear the machine is driven around the column in a horizontal position. It therefore has a universal movement and the operator can make a horizontal or a vertical cut, at his option, or can make any adjustment at will without changing the mounting.

The bit-chucks used with the Whitcomb machine are the result of a series of careful tests. They are made of steel and hardened, while the bits, or cutting-points, are made of tool-steel. The chucks are held to the extension-bars by means of a tapered socket, and the bits are held in the chucks by the same method. The chucks are made to accommodate three, four, five, or seven bits. Each machine is equipped with four extension-bars, the shortest of which is 1 ft. long and is used for starting the cut. The other bars are 2, 4, or 6 ft. long, respectively, for making and deepening the cut.

The post machines are not recommended for making an undercut on the floor, as they cannot be set for making a cut exactly at the bottom of the coal. The claims for efficiency are made for special work in mining out dirt bands or for shearing-work in driving entries, Fig. 5. They can, like the other machines of the same type, be used in pitching-beds, and can be set to conform to the pitch and to make the cut on the pitch.

3. *The Ingersoll-Rand Radialaxe.*—In a circular descriptive of this machine, recently issued, the Ingersoll-Rand Co. begins with the statement that:

In approximately horizontal coal-beds the radialaxe will not undermine as rapidly as the punching-machine, reference being made, of course, to the New Ingersoll puncher. It (the radialaxe) will not cut close to the bottom and

makes a narrow, slot-like cut, both of which make unfavorable comparison with the tapering cut made at the bottom of the coal by the puncher-machine. There are, however, five classes of work to which the radialaxe is distinctly adapted. These are:

1. Undercutting in pitching seams. The radialaxe is adapted for undercutting in beds pitching from 15° to 40°. Undercutting of any kind is usually not attempted in beds pitching more than 40°.

2. Shearing in any seam, and particularly in entries, as shown in Fig. 6. While shearing has not yet assumed in America the importance it merits, it is recognized as being almost as important as undercutting in that it tends towards the lessening of blown-out shots and the dangers therefrom. In Great Britain this element of danger was recognized a number of years ago and shooting from the solid is prevented by act of Parliament.

3. Mining near the middle of the seam. It is especially useful in driving entries where there is a constant accumulation of water which makes mining in or near the bottom of the bed an impossibility.

4. Cutting out bands of fire-clay or slate softer than or of about the same hardness as the coal, or mining under or above such bands, so that the dirty material can be loaded out without mixing with the coal.

5. Entry-driving. Entry-driving calls for a rapid and reliable shearing-machine possessing the qualities claimed for the radialaxe.

The radialaxe is in essentials a long-stroke rock-drill provided with a method of mounting especially adapted for coal-mining work. In appearance it is similar to the standard Ingersoll-Rand rock-drill. In fact, it is simply a modification of one of the company's standard drill types. It has long through bolts holding front and back heads in place, screwed home on the Sergeant patent flat steel cushion spring, and is fitted with the Sergeant release rotation, which permits throwing out the rotation in shearing or undercutting and throwing it in when the machine is to be used as a rock-drill. The piston-rod, instead of terminating in a chuck of the ordinary type, ends in a taper fitting into an extension which also receives the steels. This arrangement dispenses with a split front head. The piston is cushioned in front by means of a special cushion-valve which entraps a portion of the air when the steel shoots forward beyond normal stroke, as when a seam or pocket is encountered.

The valve-movement is that known as the Sergeant 52 type, having a means by which wear and consequent leakage past the valve can be avoided by proper adjustment of the regulating-screws at either end of the valve-chest. This is a most valuable feature, providing means by which the valve-action and consequent operation of the machine can be maintained at the point of maximum effectiveness.

The primary support is a standard double- or single-screw column, which can be furnished in various lengths from 2 ft. upward. With this column a very short arm is supplied, upon which the radialaxe is mounted for shearing. A safety clamp, or collar, on the column below the arm permits the latter to be swung to any position without sliding down. A worm-wheel sector-attachment is mounted on the arm for shearing, or directly on the column for undercutting or cutting out bands. A saddle with two cone cups is mounted on the sector-sleeve. Extending backward from this saddle is a bracket supporting a steel worm meshing with the sector-teeth, on the spindle of which is mounted a hand wheel. The guide-shell, similar to that of the ordinary rock-drill, carries a feed-screw, standards, and a hand-wheel on the screw. By the manipulation of this latter the radialaxe is fed forward or backward in its guide-shell. By the manipulation of the other wheel on the worm-spindle the radialaxe is swung radially around the axis of the arm or column, as the case may be. Several points are to be noted in this mounting.

The radialaxe rests in a cone-cup and can be released by a bolt, instead of being rigidly connected to its saddle. This permits a subdivision of weight in moving or setting up, and makes easier the handling of the machine.

The worm-gearing is outside the radialaxe instead of between it and the column. This reduces the turning-moment when mounted on the arm, tending to throw the machine out of line when running.

The worm-sector is of very large diameter and the worm of small pitch, making the swinging of the machine very easy on the operator without the use of intermediate gears.

The main trunnion, which is exposed to the full shock, is of very large diameter and length and shows none of that sear which results in lost motion, unsteady mounting, and difficult cutting.

A standard double- or single-screw column is used, which may also be used with an ordinary rock-drill.

Extensions for the radialaxe are made of the best octagon drill-steel with tapers at either end fitting the extension bushing and the bits. They are furnished in different lengths covering the range of operation of the machine.

The bits of the radialaxe are of the five-pronged (seven-pronged) pattern. The cutters are forged from a high-grade tool-steel and tapered to fit holes in the chuck which fits on the taper end of the extension-steels. The cutters range outward from the center, giving the necessary clearance in cutting. The construction is exceptionally compact and powerful, and the individual cutters are easily forged or sharpened by the average blacksmith.

III. Electric Punchers.

As previously stated, numerous attempts have been made to apply the electric current to machines of the percussion type, but so far, at least, as applied to the puncher type of coal-mining machine, the inventions have been successful only in connection with the use of compressed air. The main objects sought have been the reduction of cost in the installation and up-keep of the piping necessary to convey the air from the compressor on the surface to the many and frequently remote working-places, and a reduction in the power required for doing the work. The problem has been solved by a combination of electricity with air; the electrical energy being conducted from the generator at the surface, by wires, to a motor forming a part of the puncher-machine. The air is compressed and used in a cylinder which is a part of the complete machine. The machine representing the embodiment of this idea as applied to coal-punchers is made by the Pneumelectric Machine Co., of Syracuse, N. Y. The Ingersoll-Rand Co. have also done something along this line in the Temple-Ingersoll machine, which involves the use of a small compressor located near the face of the coal, the air being conducted to the machine by means of short pipes or

hose. This arrangement, however, has found a better application in connection with the operation of rock-drills. This Temple-Ingersoll arrangement has been described by W. L. Saunders, in his paper Electric-Air Drills.[5]

The Pneumelectric Coal-Puncher.—In this exclusively coal-mining machine the electric motor and the percussive apparatus, actuated by compressed air, are contained within the same machine. The air used in striking the blow is compressed at each stroke of the piston. The piston compressing the air is operated by the electric motor, which, as shown in Figs. 7 and 8, is an inherent part of the machine. In presenting the claims of this machine to the consideration of the mine-operators, the Pneumelectric Machine Co. gives the following distinctive points:

1. No hose or flexible shafts are required.
2. The electric power is supplied direct to the machine by a cable.
3. It is operated on the circuits commonly found in coal-mines.
4. It has a large capacity for work at low cost of operation and repair.
5. Shock to the machine is prevented by air-cushions.
6. The piston of the percussion-tool recovers in its forward motion independent of the compressing-piston.
7. It can be operated in any coal-mine where the ordinary punching-machines are employed.

The description of this machine, given below, has been supplied through the courtesy of the company:

The pneumelectric machine, Fig. 7, has been built to meet the demand for a percussive tool operated by electricity. That it has been possible to overcome successfully the difficulties met with heretofore is due entirely to the fact that electricity alone has not been relied upon. By combining the well-established methods of applying electricity with those of applying compressed air, the desired results have been brought about.

The elements employed in producing these results are: a motor, gearing, a cylinder, pistons, and piston-rods.

These are combined into one compact, self-contained machine which is so constructed that it closely resembles the compressed-air machines which have been on the market for many years. The dimensions and weight are practically the same in both machines.

The motor, which is of the ordinary series-wound type, possesses no unusual features tending to make it complicated or requiring special generators for furnishing current; but, being designed along standard lines with particular reference to this machine, it presents features which insure the greatest simplicity and highest efficiency.

The gearing is most simple and provides a straight-line motion for the operation of the piston-rod, without requiring a complicated arrangement, and reduces the space occupied to a minimum.

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[5] *Trans.*, xxxviii., 472 to 481 (1908).

The cylinder is a plain casting of the ordinary type. The only unusual feature is that it contains two pistons instead of one. These pistons have no mechanical connection whatsoever.

The operation of the machine is as follows: The rear piston, the one driven by the motor, compresses air on its backward stroke, the front piston, which carries the tool-rod, following simultaneously. The blow, which is the desired result, is obtained by permitting the air behind the motor-driven piston to pass to the front by means of open ports, and act, under expansion, upon the front piston. This piston, being free to move under the force of the expanding air, rushes forward at great speed and causes the tool to strike the coal with force approximating 1,800 pounds.

During the time the compressed air is passing from the rear to the front of the compressing piston the front piston moves forward and the rear piston continues to compress air to the end of its stroke, thus maintaining a continuous supply of compressed air.

Should the pick not strike the coal the force of the blow is taken by the cushion of air at the front of the cylinder. This is so perfect that the machine is entirely relieved of any destructive vibration or jar.

The "kick" of the machine, which is necessary to permit proper handling, is obtained through the action of the pistons on the air between them.

The motor-frame and gear-case cover are made in one piece, which is of cast-steel. In this way great strength and minimum weight are secured and the number of pieces is very much reduced. This is designed with particular reference to space occupied, and at the same time a very high efficiency of the motor is obtained.

The starting- or controlling-box is mounted on the motor. It is so constructed that all movable parts are protected. The lever carrying the movable contact is contained in the box and is operated by a lever from without. A compact arrangement for connecting the cable to the starting-box has been specially designed and permits of attaching and detaching the cable instantaneously, and at no time necessitates exposure of any of the terminals.

IV. Chain-Breast Machines.

In the chain-breast machine the method of attacking the coal is as radically different from that employed in operating the punch as sawing a log is different from chopping it. It derives its name from the fact that the cutting-tools are bits fastened to the outside of an endless chain which travels on a frame or arm extending from the body of the machine and cuts a channel in the bottom of the coal. The cut made by the chain-machine is different from that of the puncher in that the latter is V-shaped, tapering to the rear, while the former is of uniform height throughout. In thick beds where the chain-machine is used, it is found beneficial to "hole out" some of the coal in front and above the cut in order that, when shot, the coal will fall out easily from the face. The principal advantages claimed for machines of this type are the rapidity with which they do the work, the small quantity of fine coal made in

the cutting, and the freedom from the racking-action to which the operator of the punching-machine is subjected.

The principles of the chain-machine and the history of its development were extensively treated in my previous paper and need not be repeated here. The improvements made in the past 10 years have been chiefly in the way of refinement, increasing the strength and improving the quality of the materials entering into their construction. They can be operated either by air or by electricity. The latter is the motive-power most generally used.

Four companies are engaged in the manufacture of chain-breast machines, namely: The Goodman Manufacturing Co., Chicago, Ill.; the Morgan-Gardner Electric Co., Chicago, Ill.; the Sullivan Machinery Co., Chicago, Ill., and Claremont, N. H.; and the Jeffrey Manufacturing Co., Columbus, Ohio.

1. *The Goodman Machine.*—This machine, illustrated in Fig. 9, was formerly known as the Independent or Link Belt machine, manufactured by the Link Belt Machinery Co. Some years ago the Goodman Manufacturing Co. was organized by H. E. Goodman and others and took over the entire mining-machinery department of the Link Belt Co. Mr. Goodman had previously been manager of this department of the Link Belt Co. The following statement regarding the improvements made during the last few years has been furnished by Mr. Goodman:

The main improvement in the Goodman machine, during the last few years, has been that of refinement; that is, making a stronger machine and improving the materials which enter into its construction. One of the special features brought out by the Goodman Manufacturing Co., in connection with its chain-machine, is the broken-back, drop-end truck, which, as shown by the illustration, facilitates unloading the machine from, and loading it on, the truck, which is claimed to be far superior to the truck formerly used. This truck is self-propelling. A cross-shaft on the truck projects into the truck-frame, while another is fastened by a removable bushing, so that its bevel-gear meshes with the bevel-gear on the mining-machine. Another special feature recently brought out by the Goodman Manufacturing Co. is the gas-tight or flame-tight construction, which minimizes any danger of igniting dust or gas when in operation. In addition to this, the following claims are made:

1. The cutting-chain and movable frame are underneath the stationary, or body, frame, which enables the machine to make a cut very close to the floor of the mine.

2. The front jack holding the machine in place transmits its thrust through the frame to the shoe supporting the other end. This is claimed to eliminate any bending stress, because the thrust is transmitted directly to the floor instead of to the frame of the machine.

3. The motor is compound, the series field of which enables the machine to cut effectively on low voltage, and gives a powerful effect on overloads.

4. Rollers are placed at the rear end, in order to reduce the amount of effort required to move the machine sidewise. These rollers are locked by the thrust of the rear jack while the machine is in cutting position

5. Both the front and rear jacks are placed in sockets which do not fall to the mine-floor while released. This materially reduces the labor required to handle the machine.

6. The machines are equipped with a safety plug—a cast-iron key of standard cross-section. This is the weakest point in the machine, and in case of overload it breaks at loads which will not overstrain the balance of the machinery, and thereby protects the gearing and electrical equipment from sudden shocks or rough treatment.

2. *The Morgan-Gardner Machine.*—The description of this machine given in my earlier paper fits in a general way the present product of the factory. The company has, however, found that it was impossible to make one machine suit all the conditions necessary to fill, and it is now building three types of breast-machines, all standard as to duplicate parts. They are designed to operate in beds as low as 3 ft. thick, and are practically the same in style, the difference being in the weight and size of parts. The weights, complete, are respectively, 2,400, 2,800, and 3,300 lb. The company is also building two types of "low-vein" machines, one weighing 2,200, and the other 2,600 lb., similar in style to the other machines except that in the smaller of the "low-vein" machines the armature runs horizontally. This machine is only 19 in. in over-all height.

Another improvement which the company has made in its equipment for its chain-machine is in a truck of the self-propelling type, which is now extensively used. In the earlier designs of trucks the custom was to use a reverse-switch on the motor, so that when it was necessary to back out of a room the direction of rotation of the motor had to be changed. The objection to this lay in the fact that if one neglected to throw the reverse-switch to its proper place when ready to make a cut under the coal, the machine, instead of going forward, would go back in the frame and would, to use a mine phrase, "run through itself," with disastrous effect upon the machine. In order to overcome this the company has brought out what is known as a "reversible self-propelling truck," shown in Fig. 10, constructed so as to be reversed in itself, thereby obviating the

reversing of the motor, which had been an objectionable feature of the self-propelling truck.

Still another improvement claimed for the Morgan-Gardner machine is the "Keystone" chain and bit. This chain is used on all of its machines and has met with excellent success. It has been adopted as the standard by the company.

The company formerly brought out an electric pick-machine, but it has been abandoned, as it was found after seven years of experimenting that it was impossible to make an electric machine do the work, unless it was so heavy that it could not be handled by the operator.

The company also manufactures a long-wall machine, but it is principally designed for the foreign trade, as the long-wall method is used in comparatively few mines in the United States.

The company has also introduced a short-wall machine for use in mines where the conditions are not favorable for the standard breast-machine. This machine is strongly built and weighs about 3,000 lb. It is pulled across the face by a steel cable, and is held against the face by a hemp rope. This machine also has met with gratifying success.

The growth of business of the Morgan-Gardner Co. is indicated by the fact that when it began manufacturing, in 1894, it occupied 2,000 sq. ft. of floor-space, which has been increased to 75,000 sq. ft. at the present time.

3. *The Sullivan Chain-Machine.*—This is a new machine for mining coal in room-and-pillar mines, and is designated as the Sullivan continuous coal-cutter, Fig. 11. The superiority claimed for it is that it acts as a long-wall machine, in that after the sumping-cut is made the machine moves along the face by its own power, cutting as it goes, and does not have to be withdrawn, moved across the face by hand, and again advanced under the coal.

The machine is mounted upon a self-propelling truck, and all of its movements, in loading upon or unloading from the truck, and during the process of mining, are made under its own power, without hand labor. After the jacks have once been set, they are not touched until the room is completely mined. Another claim made for this machine is that it occupies less than half the space required for the standard breast-machine in

front of the coal, and can therefore be operated in mines when the condition of the roof requires that timbering be kept up too close to the face for the breast-machines. The Sullivan continuous cutter will work in a room where the props have to be kept within 6 ft. of the face, whereas from 11 to 14 ft. clearance is required for the breast-machines.

Another advantage claimed for this machine is that it will follow a rolling-bottom readily, maintaining a cut parallel with the floor. It is also claimed that on account of the narrow cutter-bar (22 in.) and its solid construction, it will cut itself free in coal which falls while being undermined. It can also be used for making break-throughs and in robbing pillars. The machine is stated to have been used across pitches of 25° and in rooms and entries driven up a pitch of 14°.

4. *The Jeffrey Machine.*—The first practical coal-undercutting machine in which the principle of percussion (as employed in the puncher type of machine) was avoided was the invention of Francis M. Lechner, of Columbus, Ohio. Mr. Lechner was fortunate in being able to interest J. A. Jeffrey, President of the Jeffrey Manufacturing Co., in his invention, and under the management of Mr. Jeffrey, the Lechner patents were developed into a successful mining-machine and a revolution in the mining of bituminous coal was inaugurated.

The earliest type of machine in which the coal was attacked on the principle of a saw rather than a pick was considerably different from that in common use to-day. In Lechner's first machine this cutting-element consisted of a revolving horizontal bar about 3.5 ft. long, carrying on its circumference cutters or bits, Fig. 12. The cutter-bar, mounted at the end of a movable frame, was rotated by a sprocket-and-chain drive from the motor, and was forced broadside against the face of the coal. The driving-chain was run so that the lower side dragged out a part of the cuttings from the slot, or kerf, and other chains provided with scraper-links were arranged to clear out the cuttings at other points along the length of the cutter-bar. At that time (1877) electricity had not reached the high state of development it has to-day, and the coal-mining machines were driven by compressed air. As is usual in the earlier stages of revolutionary inventions, there were objectionable features in the cutter-bar machine, and in

studying how to overcome them the type of machine known as the chain-breast machine was developed, and with the advent of this great improvement the cutter-bars became obsolete and were relegated to the scrap-heap. At about the same time electricity was substituted for compressed air for power-transmission, as shown in Fig. 13.

The progress made in the Jeffrey chain-machine since my former paper was presented has been in the efforts to secure the highest degree of perfection in the quality of the materials entering into the construction of the machines, and to insure in this way the highest quality in the finished machines. During the 30 or more years of its existence the Jeffrey Manufacturing Co. has kept a close watch on its various products until now there is a written history of the success or failure of the various materials used and of the proportions going into the machinery manufactured. Rigid inspection and tests, both chemical and physical, are made of all the raw materials purchased, such as the steel billet out of which an armature-shaft is forged. Similarly, other metals, the coal, coke, foundry-iron, etc., undergo careful inspection and analysis at the hands of competent specialists, and if found below the limits required, are rejected. Finally, when all parts are assembled in the completed machine, such tests and inspection are given as will discover any defects in workmanship, material, or design which may have escaped previous notice.

As the general construction of the chain-machine has been previously described and is well known, it is not considered advisable to enter into any detailed description of the Jeffrey machine at this time. There are some special features, however, which might escape notice and which may be briefly adverted to. For instance, the company is calling particular attention to the bits used in the cutting-chain. These bits are of two general sorts, chisel- and pick-pointed. The chisel-pointed bit is more commonly used, especially where hard or tough material is to be cut. Where the material is dry, soft, and brittle, the pick-pointed bit is sometimes used exclusively, and in other cases a combination of pick- and chisel-pointed bits in the same chain is used. In Jeffrey coal-cutters three general arrangements of bits are made, known as three-bit, four-bit, and five-bit. Fig. 14 clearly shows how the bits are

brought into contact with the material being cut. The four-position chain is most commonly used, the five-position chain being called for when the material being cut is exceptionally hard and tough, and the three-position chain being used where the material is either soft, brittle, or both. When both pick-point and chisel-point bits are used in the same chain, it is best to set the pick-points in advance of the chisel-points about 0.25 in. so that they will score the coal ahead of the chisel-pointed bits, making it easier for the latter to remove that which is left. For the most efficient and economical operation of the machines, different materials require different arrangements of bits and different speeds of feed. For instance, with a very dry, brittle coal, free from sulphur, pick-pointed bits can be used with a fast feed ahead, so that a cut 6 ft. deep may be made in 3 min., while on the other hand, hard, wet coal containing considerable sulphur requires the use of chisel-pointed bits and a considerably slower feed ahead, so that it may require from 5 to 6 min. to make a 6-ft. cut. The exact arrangement and style of bits and the best speed of feed can be readily determined by a few trials, and when decided upon, care should be taken to see that the machine-runners set the bits in the manner found to be most desirable.

The standard Jeffrey machines are known by a purely arbitrary nomenclature, such as the 16–A, 17–A, 21–A, etc., which represent different sizes and strengths in machines adapted to varying conditions in the mine. The 16–A machine, for example, is only 20 in. high and can be operated in a bed of coal but 30 in. thick. A larger and more powerful machine is the 17–A, which can be used in a 38-in. bed of coal, while the 19–A machine, the strongest one, may be used in a bed of coal 3 ft. 6 in. thick.

While electricity is now used almost exclusively in the operation of chain-machines, compressed air is not entirely abandoned, and one of the machines made by the Jeffrey Co. is the 16–D machine, in which compressed air is the motive-power.

A considerable number of the machines turned out by the Jeffrey Manufacturing Co. are exported, Great Britain being the principal foreign market. The British rules provide that for the operation of machinery by electricity in gaseous mines, inclosed, flame-tight motors, switches, and accessories must be

used. By "flame-tight" is meant such an inclosure that, should an electric arc or explosion of gas occur within that inclosure, there would be no ignition of gas in whatever form or proportions it might exist immediately surrounding that inclosure. The Jeffrey Manufacturing Co. immediately began the manufacture of flame-tight motors, Fig. 14, and these have passed the inspection of the experts of the British government.

Most of the machines exported are long-wall machines, which differ somewhat from the chain-machine in that the cutting-bits, instead of being carried on a chain, are carried on the periphery of a wheel operating from the side of the machine. These machines are used to some extent in the United States, but not to the same extent as in European countries. The Jeffrey Manufacturing Co. manufactures also a side-cutting machine which it designates as Class 26–A. The difference between this machine and the standard breast type is that the side-cutter makes its initial cut at one end of the working-face, and then continues the cut across the room before it is again withdrawn. The machine unloads itself from the truck, in the first place, and pulls itself across the room by its own power. The principal advantage of the side-cutter is that it occupies less space in front of the coal and can be used in places where it is necessary to keep the props set close to the face, Fig. 15.

All of the equipment of the Jeffrey machines, including the motors, is manufactured in the shops of the company.

5. *The Lee Machine.*—This machine is of a somewhat different pattern from the standard types of undercutting-machines. It has already been described by H. Foster Bain in his paper.[6] In this machine the cutting-tool consists of a rotary bar extending from the side of the machine, which is set with cutting-bits spirally arranged around the circumference in such manner that the revolutions, in addition to cutting the coal, also withdraw the cuttings from the kerf. These machines have had rather a limited application, but they are in use in several mines in Illinois, Iowa, and Colorado. The type of the machine at the present time is practically the same as that described by Mr. Bain.

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[6] *Trans.*, xxix., 474 to 482 (1899).

Pyritic Smelting in Leadville.

BY CHARLES H. DOOLITTLE, E.M.,* SALT LAKE CITY, UTAH, AND ROYAL P. JARVIS, E.M., PH.D.,† KNOXVILLE, TENN.

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The following notes are contributed, not with the idea of offering a complete history of the development of this very important process as applied to the Leadville district, but with the hope that the practice, as it was carried out in the Bi-Metallic Smelter, may still be of interest to others.

The Bi-Metallic Smelter, at Leadville, Colo., was the first to make a commercial success of the process described in the following pages, which consisted essentially of smelting raw silver- and copper-bearing sulphide ores in blast-furnaces to form a matte, which was generally re-smelted with siliceous ores for a second high-grade matte.

Robert Sticht, in his excellent monograph on the subject,[1] has outlined briefly the plant belonging to this company. He says, in effect:

> The Boulder, Mont., enterprise set the example for others with whom the patentee (W. L. Austin) had been negotiating, and the old La Plata lead-smelter at Leadville, Colo., was fashioned into a crude, temporary experimental plant for the Bi-Metallic Company, and the feasibility of the process demonstrated by a trial run. Subsequently, the plant was considerably improved, and, being supplied with very favorable ores, continued to run without interruption from August, 1892, to June, 1893. In September, 1893, smelting was resumed in a completely remodeled establishment, some $200,000 having been expended upon the new work.

Thus, aside from the Montana experiments, which, under the direction of W. L. Austin, began as early as 1885 at Toston,[2] and continued with indifferent success for six or seven years, the Bi-Metallic plant at Leadville may be said to have

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* Formerly Superintendent, Bi-Metallic Smelting Co.

† Formerly Chief Assayer and Chemist, Bi-Metallic Smelting Co.

[1] *Modern Copper Smelting*, E. D. Peters, 7th ed., p. 415 (1895).

[2] *Trans.*, xvi., p. 259 (1887–88).

been the first and most successful pyritic smelter in the country. The great success which attended the Leadville plant did much to stimulate endeavors in other localities, and enterprises modeling their practice more or less upon that of the Bi-Metallic sprang up in many places. Many, of course, were doomed to failure, owing usually to insufficient ore-supply. It was found that ores rich in sulphur could be successfully smelted without a preliminary roast, and that the resulting low-grade matte made an excellent base to flux a considerable percentage of dry siliceous ores, and thus enrich the matte. At the same time a high treatment-rate could be charged on siliceous ores. In only a few instances, however, were the conditions present to insure financial success.

It is with the plant as constructed in 1893, and the principal improvements made later, that we shall deal.

Description of Plant.

The plant consisted of three blast-furnaces. No. 1 was 36 by 163 in. at tuyeres, 48 by 175 in. at feed-floor, and 6 ft. 3.75 in. from center of tuyere to feed-floor. No. 2, as originally built, was 30 by 140 in. at tuyeres, and of the same height as No. 1, but with this important difference, that the jackets had straight sides, and without bosh. In 1898 this furnace was entirely reconstructed, with the following dimensions: 36 by 215 in. at tuyeres, 48 by 215 in. at feed-floor, 6 ft. 6.75 in. from center of tuyere to feed-floor. No. 3 was 36 by 175 in. at tuyeres, 48 by 175 in. at feed-floor, and 6 ft. 3.75 in. from center of tuyere to feed-floor.

There were two rows of cast-iron water-jackets. The lower row of jackets was supported on a cast-iron bed-plate 2.5 in. thick, which was in turn supported by eight or ten jack-screws. The bed-plate was protected inside by an 8 or 9 in. layer of fire-brick. The distance from the center of tuyeres to the top of the cast-iron bed-plate was 21.75 in. in all cases. This left an interior crucible about 13 in. deep from the tuyere-center to the top of the brick lining. All furnaces were provided with sumps, lined with fire-brick, attached to the middle of the long side of the furnace. The nose or end of the sump was protected by a dam of clay, which was renewed from time to time as it was cut out by the stream of slag and matte.

The stream of matte and slag discharged continuously through the sump into small rectangular settling-boxes, the largest of which was 5.5 ft. long, 3 ft. wide and 2 ft. 9 in. deep, and lined with a single 4-in. layer of fire-brick. From these settlers the matte was tapped out into pots, and the slag overflowed continuously, passing through two large conical pots in succession, and finally into the pots which were transported to the dump. Each slag-car carried two conical pots, mounted in such a way that they could easily be tilted sidewise when full, and righted again by means of a bar inserted in a boss on the side. The slag was hauled from the furnaces to the dump by a horse.

The brick superstructure of all three furnaces was carried on the usual four-pillar and girder-system, and built to an unusual height. A single sheet-iron down-comer, supported on cross-beams and railroad iron, was connected to a common V-bottomed sheet-iron dust-chamber, carried on wooden bents, and 10 ft. above the level of the charging-floor. A narrow-gauge tram-track was laid beneath the flue, and the flue-dust was drawn off into ordinary mine-cars and carried to the briquetting-department. Outside the furnace-building the iron dust-chamber was connected with one of brick, which was provided with suitable gates, so that the furnace-gases could either be diverted to the dry flue-dust catching and slime-recovery towers or conducted directly to the main chimney. The tops of the furnaces, at feed-floor level, were covered by two sliding doors on each side, with counter-weights.

Slime-Plant.

At this plant a unique system of recovering the condensible constituents in the furnace-gases was devised by the late Franklin Ballou, manager of the plant. The scheme consisted essentially of drawing the dry, hot gases from the main brick flue-chamber, noted above, down into a round brick dust-chamber, provided at the bottom with suitable hoppers to draw off the flue-dust. The gases were then drawn through two Sturtevant exhaust-fans and forced through a pair of brick scrubbing-towers, the interior of which was filled with a series of V-shaped tiles supported on lead-incased I-beams. The tops of the towers were covered with a shallow lead-lined reservoir,

perforated with a number of 0.5-in. lead pipes, with the upper ends of the pipes slit for about 1 in., through which water trickled into the pipes and fell upon the upper tier of tiles. The upper ends of the lead pipes were open, so that in case the narrow slits became filled with silt and other matter, water could still pass into the tower when the level in the reservoir rose to the tops of the pipes. The towers being divided into halves by a vertical partition, the gases entered the bottom of one compartment of each of the towers, passed upward, then down through the second compartment and into the main flue-chamber leading to the stack. The main stack was originally of iron, but the moist acid fumes soon corroded it completely, and it was replaced by a wooden one. This wooden stack was 120 ft. high, 10 ft. in diameter, and octagonal in section. A second and auxiliary iron stack, 60 ft. high and 10 ft. in diameter, was built later, to be used in case of accident to the fans or the slime-plant when the hot furnace-gases could not be drawn through the towers and cooled before leading them to the wooden stack. Before the auxiliary iron stack was constructed it was sometimes necessary to divert the hot furnace-gases to the wooden stack, but a careful watch was always kept of the temperature at the stack. During the six years the wooden stack was in service it became necessary to reline it once. This was done shortly before the plant was finally closed, in the spring of 1900.

The Sturtevant exhaust-fans were driven by the main engine, situated nearly 400 ft. from the fans; and the power was communicated from the main line-shaft by belt and pulleys to a second line-shaft running behind the furnaces; then from this second line-shaft to a third in the same manner. Thus with the many shafts, and belting, the power required to drive the fans was always a serious matter. Probably 40 per cent. of the power of the engine was absorbed in driving them. The product recovered in the iron dust-chambers and first tower was known as "flue-dust." Its composition varied considerably; but the following is an average month's analysis during the last campaign: silver, 14.2 oz. per ton; gold, 0.12 oz. per ton; copper, 2.3; silica, 15.6; iron, 38.4; manganese, 0.5; zinc, 7.1, and lead, 3.5 per cent. Daily, samples of flue-dust were assayed for silver and lead.

The product of the scrubbing-towers was known as "slimes," and was recovered by conducting into suitable settling-tanks the water flowing from the bottoms of the towers. These rectangular wooden tanks were arranged in pairs, to enable one set to be cleaned while the other was filling. Scrap-iron was placed in the bottoms, and a considerable amount of metallic copper was recovered in this way. The composition of the slimes varied between wide limits, depending on the amount of slime and flue-dust in the charge, the content of lead in the ores, the degree to which the furnaces were crowded, etc. The slimes were essentially a lead-product, and the object was to recover the lead (with always more or less silver and gold), and sell it to the lead-smelters. The following analysis is a month's average: silver, 16.0 oz. per ton; gold, 0.11 oz. per ton: copper, 2.4; silica, 15.9; iron, 34.5; zinc, 7.0; manganese, 0.4, and lead (dry assay), 6.3 per cent. It is to be noted that this analysis represented a period of very fast driving. Assays of lead were sometimes obtained as high as 30 per cent., and assays of from 15 to 30 per cent. were made for days at a time. The silver-content never appeared to be directly related to the percentage of lead in the slimes. The slime taken from the bottom of the wooden stack, and from the flues beyond the gossage-towers, was found to be higher in lead than that normally recovered in the gossage-towers.

The flue-dust removed from flue-chambers and towers ranged from 3.5 to 7 per cent., and the slimes from 2 to 4 per cent., of the ore smelted. During February, 1900, the flue-dust was 6.5, and the slimes 3.5 per cent., of the ore smelted.

The low-grade lead-slimes, such as those corresponding to the analysis above given, were mixed with flue-dust in a pug-mill, with the addition, sometimes, of more or less lime, and run through a briquetting-machine. This machine was similar in principle to the ordinary mud-brick machine, except that the strips of mud forced out were not rectangular, but round, and were not cut off by wire cutters. An endless belt carried them away from the machine, and they were picked off by boys, and piled on wooden crates or racks, which were stacked up in kilns to dry. The kilns were simply small wooden rooms arranged in groups within the main smelter-building, and furnished with hot air blown by a Baker blower through

a stove which had originally been designed to heat the blast for the furnaces, and which consisted of a series of cast iron U-tubes, arranged in a brick furnace. The hot air from this stove was conducted through sheet-iron pipes beneath the floors of the kilns,—a dangerous arrangement, since both floors and kilns were constructed entirely of wood. One morning in the summer of 1897, the kilns were discovered to be on fire. After a stubborn fight the fire was finally put out. After this it was found desirable to remodel slightly the floors of the kilns, and to watch more closely the temperature of the air admitted to them. While this plan was a cheap one, and the location of the kilns within the main furnace-building made it convenient to handle all the flue-dust products, it was highly perilous and objectionable.

Power-Plant.

One 450-h-p. Hamilton Corliss engine, with 24- by 48-in. cylinder, run at 65 rev. per min., steam-pressure 90 to 100 lb., furnished all the power. Four 100-h-p. tubular boilers supplied steam. It was found that this battery of boilers was crowded to its limit, and at best afforded no reserve; and another unit of 100 h-p. was added in the spring of 1900. Up to the summer of 1898, two No. 7 Roots blowers furnished all the blast to the three furnaces. It was found that after reconstructing and enlarging No. 2 furnace two blowers were not supplying enough blast, and accordingly another No. 7 Roots blower was installed. All blowers were belt-driven from the main line-shaft, each blower being driven by two belts. An attempt was made in 1897 to supplement with a Sturtevant centrifugal blower, or fan, the quantity of air delivered to No. 3 furnace, but this was abandoned. It may be noted here that No. 3 was the furnace usually run on the concentrating-charge, and the idea of coupling a fan to the furnace was to supply a larger volume of air, and obtain thereby a higher ratio of concentration. The one Corliss engine had thus to supply power for the three blowers and the two exhaust-fans as a constant day-and-night load, and in addition the briquetting-plant (day-shift only, but a severe and rapidly-fluctuating load), slag-elevators, electric-light plant (night-shift), sampling-mill, and a small machine-shop.

Receiving the Ore.

The plant was a three-plane plant, with furnace-floor, feed-floor, and receiving- or sampling-floor. Much of the ore received at the smelter, except during the last year's operation, was delivered in wagons. These heavy four-mule ore-wagons, familiar to all old Leadville residents, would come, as many as a dozen at a time, and each wagon was weighed and unloaded on to a large platform covering a number of bins beneath. If the wagons were being received at a somewhat slower rate, the ore was frequently unloaded directly into the ore-bins below, a tenth being usually reserved as sample. In case it was all unloaded on the platform, the tenth or other fraction was cut out when the ore was shoveled into the bins by the yard-men. While this arrangement would be awkward and inefficient for a plant receiving all products in cars, the unloading-platform was in fact admirably adapted for the purpose in hand. Three narrow-gauge tracks were carried over the tops of the bins on a level with the wagon-platform, and much of the siliceous ore and limestone was unloaded directly from cars. After the completion of the narrow-gauge high line to the Ibex (Little Johnny) and other mines of the district, but little ore was received in wagons. A narrow-gauge track was carried to the level of the furnace-floor, and the high-grade lead-slimes and shipping-matte could be loaded from this level. In case of accident to the slag-and-matte elevator, these products could be carried to the higher level and unloaded into bins on the feed-floor. All ore-, flux-, and fuel-bins were on the level of the feed-floor. Considerable excavation was necessary to provide space for the bins, but not a shovelful of the furnace-charge was exposed to the weather, and the heavy snows of the region could not interfere with the furnace-operations.

Sampling.

The tenth cut out from the original lot was crushed in a Blake crusher, and usually a tenth or a fifth of this sent to a pair of rolls, the balance going back to the main ore-bin, while the rolled ore was coned on a coning-floor, and the sample reduced to about 100 lb. and passed through the finishing-rolls. Finally a sample of about 10 lb. was sent to an Englebach sample-grinder, and from this a sample weighing

about 21 oz. was bucked down to pass through a 100- or 120-mesh sieve, and the usual buyer's, smelter's, and umpire samples were made up. On many ores a duplicate sample was taken, beginning at the point where the first coning and quartering was made. Moisture-samples were taken at the time the ores were unloaded, kept in sealed cans until the lot was closed, when 50 oz. was weighed on a moisture-scale, dried in a steam drier, and again weighed to determine percentage of moisture. The portions cut out for samples were reserved in suitable bins in the sampling-department until the lot had been assayed and settled for. All sampling-operations were carried on by hand, no automatic or machine samplers ever having been installed.

Assay-Office, Laboratory, etc.

The assay-office and laboratory and general offices were housed in a small single wooden structure which had never been changed from the time the La Plata Co. built it.

Smelting Practice.

In the earlier period of smelting the aim had been to make a much higher ratio of concentration, and produce a matte of much higher grade; and mattes running several thousand ounces in silver had been made. It was found, however, that a better saving was made with lower ratios, and accordingly the matte from the concentrating-charge was rarely made higher than 300 oz. per ton. The copper-content of course depended upon the amount of this metal in the charge, but during the last three years did not usually run less than 14 or much above 23 per cent. This grade of matte was always the result of a second smelting, the first or green-ore charge giving a matte very low in both copper and silver. Regular furnace-calculations were rarely made. For the green-ore charge, the arithmetical mean of the net ore in the charge was taken, and the charge was proportioned in such way that the average would show from 14 to 20 oz. of silver per ton, from 2 to 3 per cent. of copper, from 3 to 5 per cent. of zinc, and an excess of metallic iron over the silica or insoluble matter of from 8 to 15 per cent. The weight of green ore was 2,600 lb. to a charge. To this was added from 225 to 250 lb. of lime-rock, 1,400 lb.

of slag from concentrating-charge, from 200 to 300 lb. of flue-dust, and 325 lb. of coke, making a total of ores and fluxes, exclusive of slag, of from 3,025 to 3,150 lb., or a total, including slag, of from 4,525 to 4,650 lb. Blank charges, consisting of fuel and slag, were given from time to time. The slag and ore were charged in layers, and the following represents a typical green-ore charge, just as the materials would be added. The second or shipping matte contained from 9 to 12 per cent. of the sulphur in the ore.

	Pounds.
Iron-Silver ore (heavy sulphide), ...	500
Coke (Crested Butte or Cardiff), ...	325
Slag, ...	500
Marion ore (iron sulphide), ...	600
Slag, ...	500
Ibex ore (copper sulphide), ...	700
Slag, ...	400
Ibex ore (copper sulphide), ...	800
Lime-rock, ...	225
Flue-dust, ...	300
Total, ...	4,850, including coke.

The coke used in this charge was from Cardiff, Colo., a medium-dense fuel with the following composition: moisture, 0.45; ash, 9.24; volatile matter, 5.66; and fixed carbon, 84.65 per cent. It was unloaded from wagons into iron tanks filled with water, to insure a thorough wetting of the coke, with the object of carrying it deeper in the furnace.

The above green-ore charge figures out as follows, assuming 65 per cent. of sulphur eliminated, which was the average maintained on the first smelting:

Green-Ore Charge, Bi-Metallic Smelter, Leadville, 1900.

Name of Ore.	Class.	Dry.	SiO_2.		FeO.		CaO.		S.		Cu.		Zn.		Ag.	
		Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Ounces Per Ton.	Ounces in Charge.
Iron-Silver	Sulphide.........	500	3.0	15.0	48.0	240	......	......	45.7	228	0.1	0.5	6.0	30	10	2.5
Marion......	Iron sulphide....	600	8.0	48.0	48.0	288	......	......	45.0	270	0.1	0.6	1.0	6	16	5.0
Ibex.....	Copper sulphide	1,500	19.0	285.0	41.0	615	.....	......	35.0	525	2.4	36.0	6.0	90	16	12.0
Limestone	....	225	4.0	9.0	2.0	4	48.0	108	......	......	......	......	......	....	......	......
Flue-dust..	....	300	14.0	42.0	45.6	137	4.0	12	20.0	60	2.5	7.5	9.0	27	14	2.0
Total....	....	3,125		399		1,284 Fe 1,000		120		1,083		44.6		153		21.5

The removal of 65 per cent. of the 1,083 lb. of sulphur leaves 35 per cent., or 379 lb., in the charge. To form matte, Cu_2S, 44 lb. of copper requires roughly 11 lb. of sulphur ($Cu_2 : S = 126 : 32 = 44.6 : 11.0$); and this amount of sulphur being deducted from 379 lb. of sulphur leaves 368 lb. which will combine with iron to form matte. This is easily calculated as follows: $Fe : S = 56 : 32 = x : 368$. $x = 644$ lb of iron. Deducting this from the total in charge (1,000 lb.) leaves 356 lb. of iron, equivalent to 458 lb. of FeO, for the formation of slag.

Assuming that all the copper, silver, and gold go into the matte, and that the sum of the copper, iron, and sulphur make 100 per cent. of matte (quite close enough for the conditions here considered), we have for the matte:

Composition of First Matte.

	Pounds.	Per cent.
Sulphur, ...	379	35.5
Iron, ...	644	60.3
Copper, ...	45—	4.2
	1,068	100.0

Silver, 21.5 oz.: equivalent to a matte carrying 40.2 oz. per ton.

Assuming that the given slag-constituents (SiO_2, FeO, CaO, and 50 per cent. of the total ZnO in charge) make 95.0 per cent. of the slag, we have

Composition of Slag for First Matte.

	Pounds.	Per Cent.	Per Cent.	Weight.
SiO_2....................	399	35.3	35.3	399
FeO....................	458	40.5	......	458
CaO....................	120	10.6	10.6	120
ZnO....................	95	8.4	......	95
Zn....................	76	......	6.6	
Fe....................	356	......	31.5	
Undetermined.......	.....	5.0	5.0	58
Total,........................		99.8	89.0	1,130

An analysis of the slag when running on the above charge, gave: SiO_2, 36.2; Fe, 32.5; CaO, 7.6; Zn, 4.0 per cent.

In the above calculation the effect of the addition of nearly

35 per cent. of slag from the concentration-charge is not taken into account. The analysis shows that the slags are lower in zinc than the one based on the assumption that 50 per cent. of the zinc in the charge goes into the slag. No doubt more than 50 per cent. is volatilized, and the matte will contain about the same percentage of zinc as the slag.

The green-ore charge shows a ratio of concentration, based on net ore smelted (2,600 lb.) and matte produced, of only 2.44 per cent.; calculated on total ore (2,600 lb.), flue-dust, and limestone, but excluding the concentration-slag, there is 1.44 per cent. of copper in the charge.

It may be added here that slag- and matte-samples were taken from every pot of slag and matte, and assays for silver were made on both every two hours. The silver-content of the slags ranged from 0.8 to 1.4 oz. per ton in the green-ore slags, and from 0.20 to 0.35 per cent. in the slags from the concentrating-charge. All of the latter slag was smelted again on the green-ore charge. Determinations for copper were made at irregular intervals, and showed generally less than 0.1 per cent. Gold-determinations were made once a week on slags, by parting all the silver buttons saved from the silver-assays. Gold-values in the slags generally ranged from 0.0040 to 0.0060 oz. per ton for the green-ore slags, and from 0.0075 to 0.0110 oz. per ton for the slags produced on the concentrating-charge. The slag from the green-ore charges was thrown over the dump.

The matte produced on the green-ore charge, which, as noted above, represents a very low concentration, was smelted with siliceous gold-, silver-, and copper-ores for a high-grade shipping-matte. An abundant supply of this class of ore was absolutely essential for the success of the process. With ample supplies of siliceous silver- and gold-ores, capable of standing a high treatment-charge, the financial success was largely assured.

The following table shows the order and amounts of ingredients charged in the concentrating-furnace:

	Pounds.
Coke,	125 to 150
Commodore siliceous silver-ore,	100 to 300
Matte, from green-ore charge,	1,000
Centennial Eureka, silver- and copper-ore,	300 to 400
Lime-rock,	100 to 125

The amounts and proportions of the siliceous ores varied somewhat. Often only Commodore ore from Creede, Colo., was available; and at times siliceous ores from Leadville were substituted. This included Ibex siliceous ores, but since much of this ore contained up to 5 per cent. of lead, it was not wanted. It may be said that of the various classes of siliceous ores used at one time or another on the concentrating-charge, the Creede ores, such as those furnished by the Commodore, Amethyst, etc., although free from copper, in general, proved the most satisfactory.

Several attempts were made to substitute Cripple Creek ores for the reliable Creede ores on the concentrating-charge, and small amounts were tried on the green-ore charges, with the result that the tonnage fell off rapidly in both cases, and the slag refused to run.

The interesting but troublesome phenomenon of the formation of magnetic oxide was sometimes noticed when, for any reason, the furnace was not making its customary tonnage. The presence of magnetite in the slag resulted in a less-perfect separation of slag and matte, a mushy matte and slag, and all the other evil results noticed by others.

The weight of low-grade matte was always kept constant, *i. e.*, 1,000 lb. on all concentration-charges.

The following shows a concentration-charge, using 300 lb. of Commodore and 300 lb. of Centennial Eureka (Utah) ore on the charge. The calculation is based upon a 75-per cent. sulphur-elimination, and on the assumption that the slag contains Cu, 0.3 per cent.; Ag, 2.0 oz. per ton; that the SiO_2, FeO, CaO, and Al_2O_3, together make up 95.0 per cent. of the slag; and that the Cu, Fe, and S make up 96.0 per cent. of the matte.

Concentrating-Charge, Bi-Metallic Smelter, Leadville, 1900.

Name of Ore.	Class.	Dry.	SiO_2.		Fe.		CaO.		S.		Cu.		Al_2O_3.		Ag.	
		Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Per Cent.	Pounds.	Ounces Per Ton.	Ounces in Charge.
Centen. Eureka.	Siliceous copper..	300	75	225	4	12	2	6	1	3	4.6	13.8	4	12	8	1
Commodore.......	Siliceous silver...	300	70	210	3	9	2	6	2	6	....	..	8	24	45	6.8
Matte	Low grade	1,000	...	....	60.2	602	...	..	35.6	356	4.2	42	...	...	40	20.0
Limestone..........	....	100	4	4	2	2	48	48	......	......	. ...	......	...	..	......	......
Total		1,700		439		625		60		365		55.8		36		27.8

We have, according to our assumptions, 1,226 lb. of slag containing 0.3 per cent., or 3.6 lb., of copper to be deducted from the total in the charge, leaving 55.8 — 3.6 = 52.2 lb. of copper to go into matte. Of the sulphur, 25 per cent. of 365 lb., or 91 lb., is left to go into matte. The 55.8 lb. of copper* requires 14 lb. of sulphur, leaving 91 — 14 = 77 lb. to combine with iron and go to matte. This 77 lb. will require: Fe : S = 56 : 32 = x : 77; x =135 lb. iron.

Subtracting this amount from the 625 lb. of iron in the charge, we have 490 lb. for slag: but since it enters the slag as FeO, the weight will be: 72 : 56 = FeO : Fe = x : 490; x = 630 lb. FeO.

We have, therefore, for the matte the following:

Composition of High-Grade or Second Matte.

	Pounds.	Per Cent.
Copper,	52.2	17.9
Sulphur,	91.0	31.4
Iron,	135.0	46.6
Other elements,	11.8	4.0
Total,	290.0	99.9

The matte will contain in addition 183.4 oz. of silver and from 1.50 to 3.0 oz. of gold per ton. The matte-fall based on ore, limestone, and matte is 17.06 per cent., and the ratio of concentration 6.6 into 1.0. This calculated matte is perhaps 2.5 per cent. higher in copper than was customarily made during the period referred to.

The slag on the concentrating-furnace, as calculated, shows: silica, 35.80; iron oxide (FeO), 51.38; lime (CaO), 4.89; alumina (Al_2O_3), 2.93, and other constituents, 5 per cent.

Smelting the above charge with 150 lb. of coke, shows a fuel-consumption for the concentrating-charge of 8.82 per cent. based on total ore, flux, and matte. With 100 lb. of coke on charge the fuel-percentage was 5.88 based on total charge. Based on net ore and flux these percentages are, for 150 lb. coke, 21.4; and for 100 lb. coke, 14.28 per cent. The ore-charge, with 325 lb. on a total charge of ore, flux, flue-dust,

* The copper which enters the slag is assumed to be in form of a sulphide (Cu_2S), and hence sufficient sulphur to cover the total copper in the charge must be provided; or in other words 55.8 lb. of copper requires 14.0 lb. of sulphur to form matte, of which 3.6 lb. of copper as matte goes into the slag.

and slag of 4,525 lb., gives a fuel-percentage of 7.19; excluding the slag, 10.4 per cent.; and excluding slag and flue-dust, but including all ore and limestone, 11.5 per cent.; and on ore alone, 12.5 per cent. It may be said, therefore, that the fuel-consumption, based on net ore alone, smelted (using 125 lb. of coke as an average figure for the concentrating-charge), was about 14.0 per cent.

The capacity of the furnaces running on the green-ore charge varied from 125 to 175 tons of net ore per 24 hr.; and per square foot of hearth-area on the largest furnace, from 2.32 to 3.25 tons of ore per 24 hours.

On the concentrating-charge a total of from 225 to 300 tons of ore, matte, and limestone was smelted per 24 hr., corresponding to from 5.14 to 6.85 tons per square foot of hearth-area based on No. 3 furnace, the one usually run.

The supervision and labor connected with the direct operation of the plant, but excluding the clerical force, was as follows:

Occupation.	No. Employed. Day.	Night.	Total.
Superintendent,	1		1
Assayers,	2	1	3
Chemist,	1		1
Blast-furnace foremen,	1	1	2
Sample-foreman,	1		1
Master-mechanic,	1		1
Engineers,	1	1	2
Firemen,	1	1	2
Charge-weighers,	1	1	2
Charge-wheelers,	9	9	18
Blast-furnace feeders, 2 on each furnace,	6	6	12
Furnacemen, } one of each for each furnace per shift,	3	3	6
Furnace-helpers,	3	3	6
Matte-tappers,	3	3	6
Pot-pushers,	3	3	6
Slag-car drivers,	1	1	2
Trackmen,	1	1	2
Stableman,	1		1
Loading slag,	6 to 10		6 to 10
Loading matte,	4 to 8		4 to 8
Sample-room,	6		6
Unloading ore and coal,	4 to 8		4 to 8
Oiler,	1		
Flue-dust and briquetting,	6 to 12		6 to 12
Carpenters,	4 to 8		4 to 8
Blacksmiths,	2		2
Blacksmiths' helpers,	2		2
Machinist,	1		1
Tower-men,	1	1	2
Handling slimes,	2 to 4		2 to 4

Those employed around the furnaces worked 12-hr. shifts; carpenters, machinists, blacksmiths, sample-men, etc., 10-hr. shifts; the slag-car drivers usually 8-hr. shifts; engineers, firemen, and tower-men, 12-hr. shifts.

Tops of Copper Blast-Furnaces.

BY N. H. EMMONS,* COPPERHILL, TENN.

(Canal Zone Meeting, November, 1910.)

AN interesting development of copper blast-furnace construction has been brought about in adapting the blast-furnace to be a "burner" for sulphuric acid making.

When the Tennessee Copper Co. first decided to construct an acid-plant, the standard type of brick-top furnace, supported by structural steel, was the only one in use at its plant. All the tests for temperature and strength of gas had been made on these furnaces, and had been satisfactory. When, however, the acid-plant was put on the flue, and the flue had to have a damper put in it to force the gas into that plant, troubles began.

The furnaces were fairly tight, and for a few months the work was quite satisfactory. Fig. 1 shows the old type of furnace-top used at the start. It was soon found that this top would not stand the temperatures, particularly when dampered back, as was necessary to force the gas into the Glover towers. The structural steel, of which the furnace skeleton was constructed, warped and twisted so badly that it was not safe to use the furnaces. This effect is shown by Fig. 2, which is a photograph of old No. 4 furnace, after the brick-work had been removed.

It became apparent that a new type of top was necessary and the engineering force was started on the design of a new furnace-top, with the result that a low top, with brick-lined flues at each end below the feed-floor, leading to the main concrete dust-chamber, was constructed. This furnace, No. 7, is shown in Fig. 3. The charging-doors are made of cast-iron sections, three sections to a door, and three doors to a side. A door to the furnace was made of cast copper, but the heat was so great that the copper bent readily, due to the jarring of its

* General Manager, Tennessee Copper Co.

operation, and proved a detriment. The top is made up of 18 ribbed copper castings, in the form of a grid, in which are laid fire-brick. The ribs extend above the brick and by radiation keep the copper cool enough to withstand the intense heat, at times from 1,800° to 2,000° F. Iron was tried in place of copper, but it would not stand the heat, and cracked and warped badly.

The whole top is hung from a structural-steel frame, supported on corner-posts, as shown in Fig. 4, which is a view of furnace No. 6, taken while undergoing repairs. The top-sections are bolted together, and asbestos is used between the joints, both to make the joints tight and to allow for the large expansion. The trouble with this furnace was that the top was too low, and when the doors were opened for charging or barring, smoke and flame shot out, making it both difficult and unpleasant to charge and bar. A good feature, however, was that the whole side of the furnace could be opened, as there were no dividers, so the charge could be dumped very readily, and exactly where it was needed.

This type of furnace-top and charge-door was very satisfactory from the stand-point of rapid repairs and convenient operation, but the top was too low and the connections to flues were too small.

Furnace No. 6 came after No. 7 and was similar to it, except that the top was made higher, giving more space for the gas. The flue-connections were enlarged and, above the feed-floor, were made of water-jackets. These, however, were not at all satisfactory, so a connection was made to the main blast-pipe and air at 45 oz. pressure was blown through the jackets, which answered very well indeed. The No. 6 top, shown in Fig. 4, gives a better idea of the construction of both No. 6 and No. 7 furnaces.

These two furnaces gave satisfactory results, so far as concerned maintaining the temperature of the gases and keeping them from being diluted with "false air." Repairs were easily and cheaply made; and the only real objection was that the gas-flues were somewhat small and low, so that flue-dust and partly-melted pieces of charge were thrown into the exit, obstructing the holes provided for the removal of flue-dust, building up accretions in the two exits, and reducing their

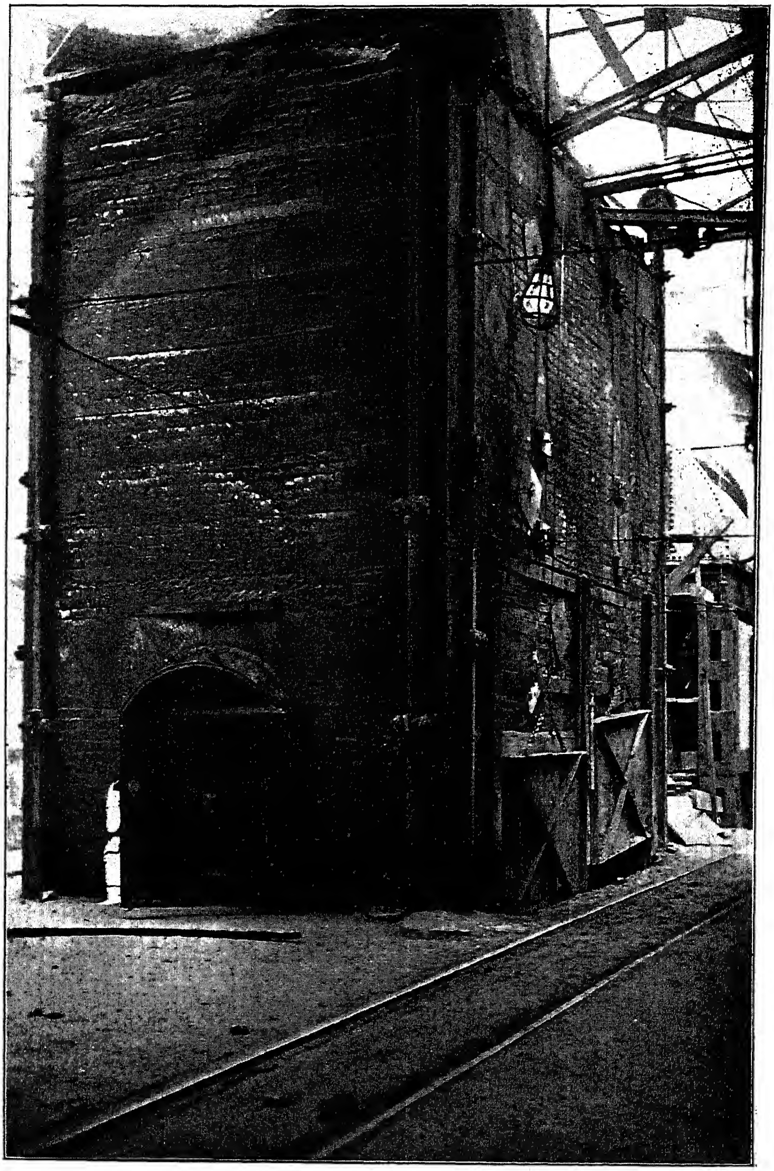


FIG. 1.—NO. 3 FURNACE. THE OLD STYLE OF TOP, REINFORCED BRICK WALLS.

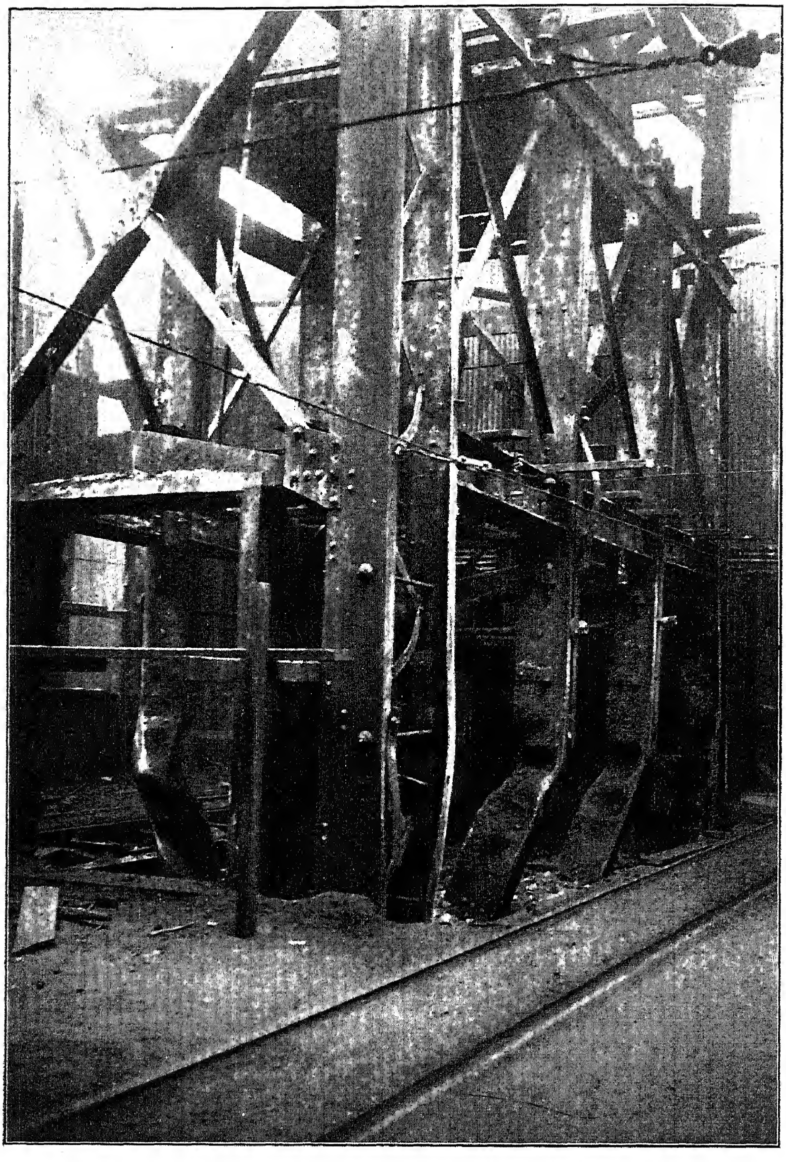


FIG. 2.—No. 4 FURNACE, SHOWING STEEL-WORK OF OLD FURNACE BEING DISMANTLED. THE CHANNEL-IRON COLUMNS AT EACH CORNER WERE PRACTICALLY SUPPORTING THE WHOLE TOP.

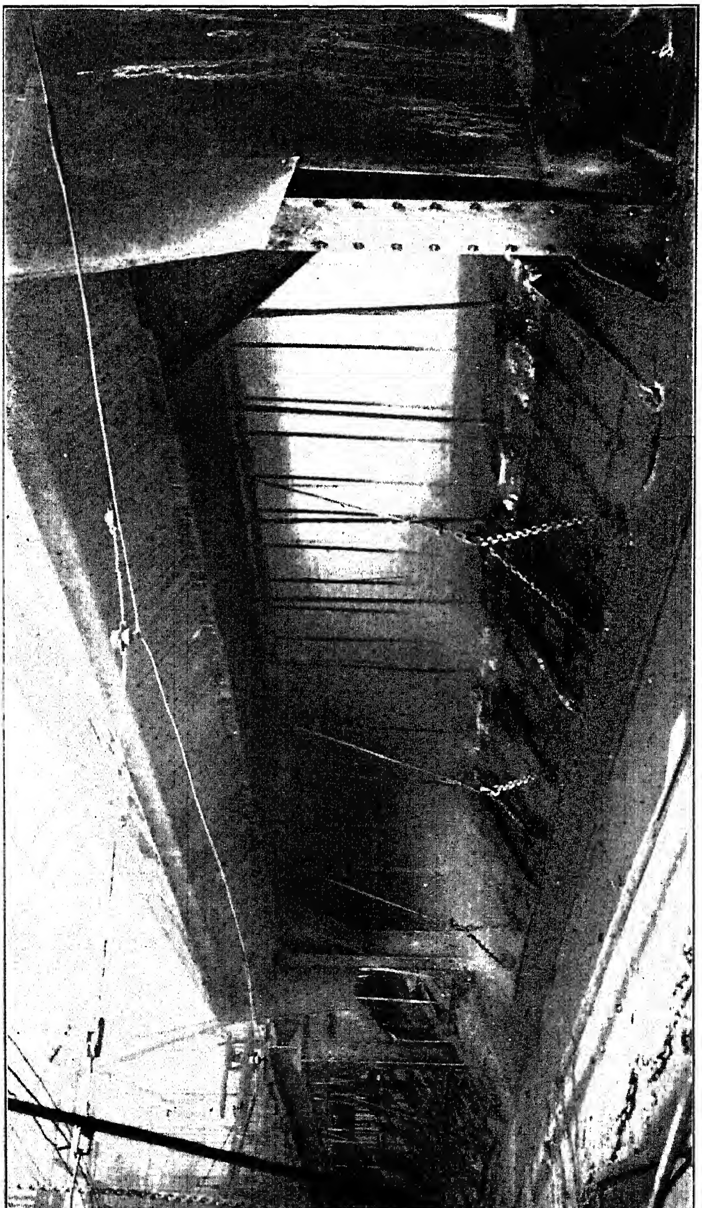


FIG. 3.—No. 7 FURNACE. THE FIRST EXPERIMENT IN LOW TOP. THE SHEET-IRON HOOD IS NOT AN ESSENTIAL PART OF THIS FURNACE. THE GASES PASS OUT OF EACH END DOWN TO FLUES, GOING UNDER THE FEED-FLOOR TO MAIN FLUE.

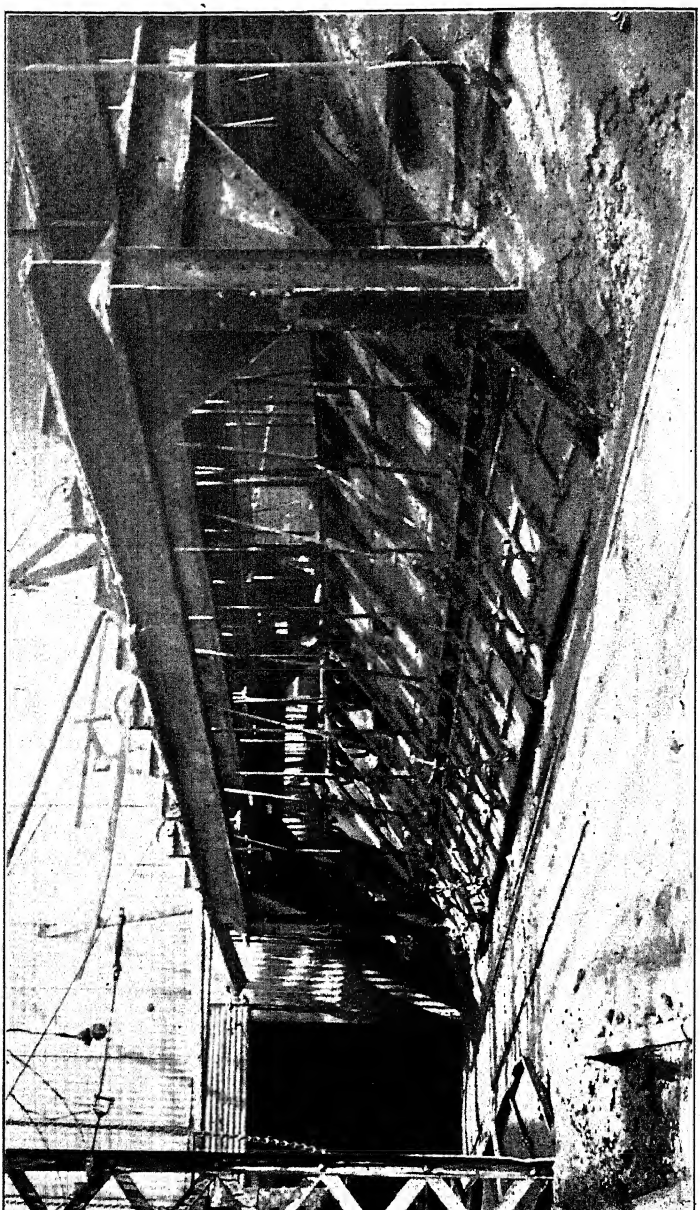


Fig. 4.—No. 6 Furnace. Undergoing Repairs, after a Severe Campaign of 24½ Days, when 13,713 Tons of Ore were Smelted under Back-Pressure; Smelting-Rate, 555 Tons per 24 Hours.

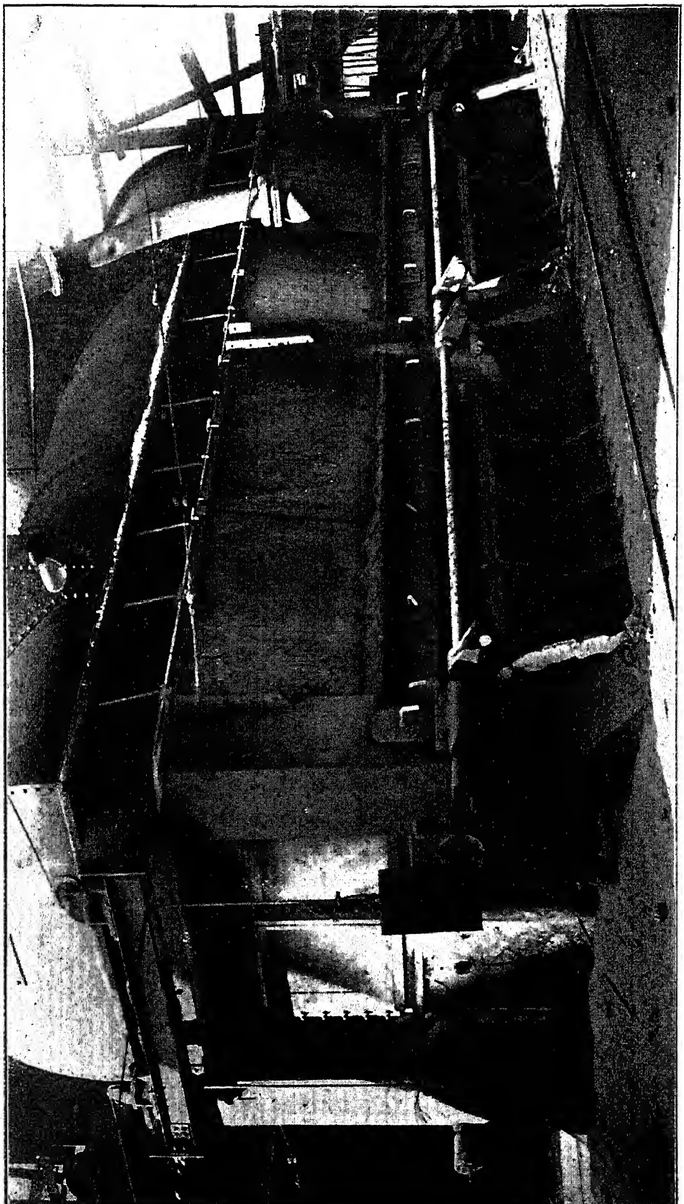


Fig. 5.—No. 5 Furnace. Cast-Iron Used Throughout from Feed-Floor to Semicircular Brick-Lined Arch. A Reversion to the Old Style of High Top.

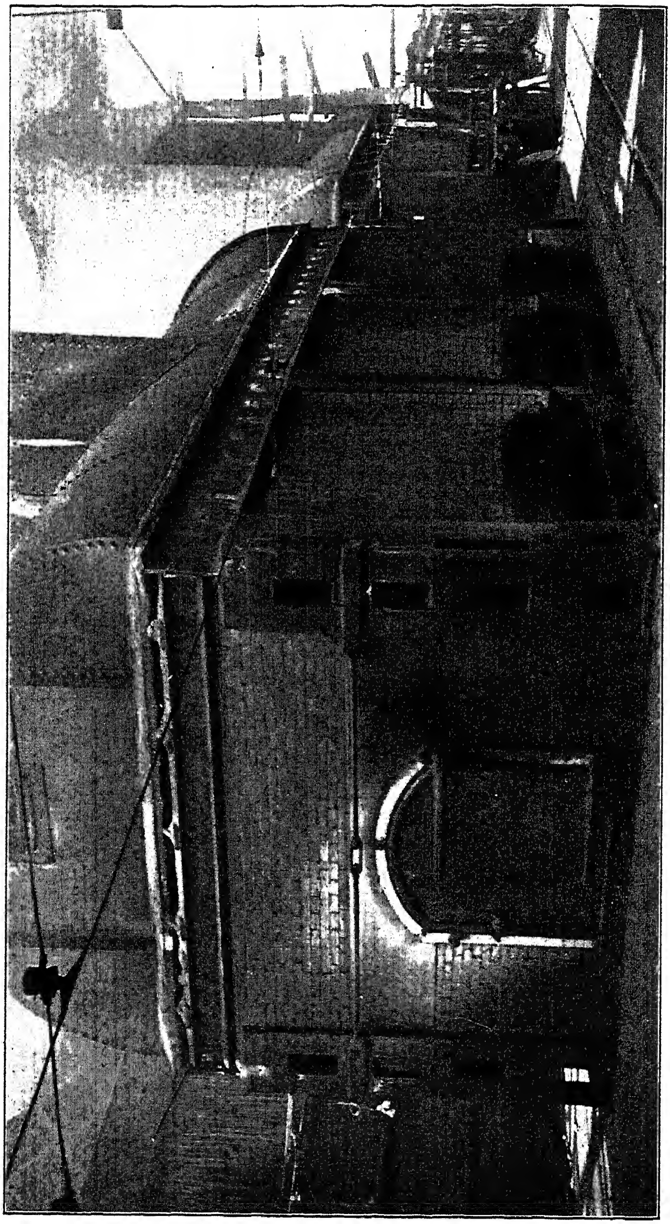


FIG. 6.—NO. 4 FURNACE (NEW). ONLY CAST-IRON AND FIRE-BRICK USED IN THIS CONSTRUCTION UP TO THE I-BEAMS SUPPORTING THE SEMICIRCULAR ARCH. THE ONLY IRON EXPOSED TO THE HEAT AND GASES IS THAT OF THE LINTELS OVER THE CHARGING-DOORS, AND LINTELS SUPPORTING THE SEMICIRCULAR ARCH. A SIMPLE, INEXPENSIVE FURNACE-TOP.

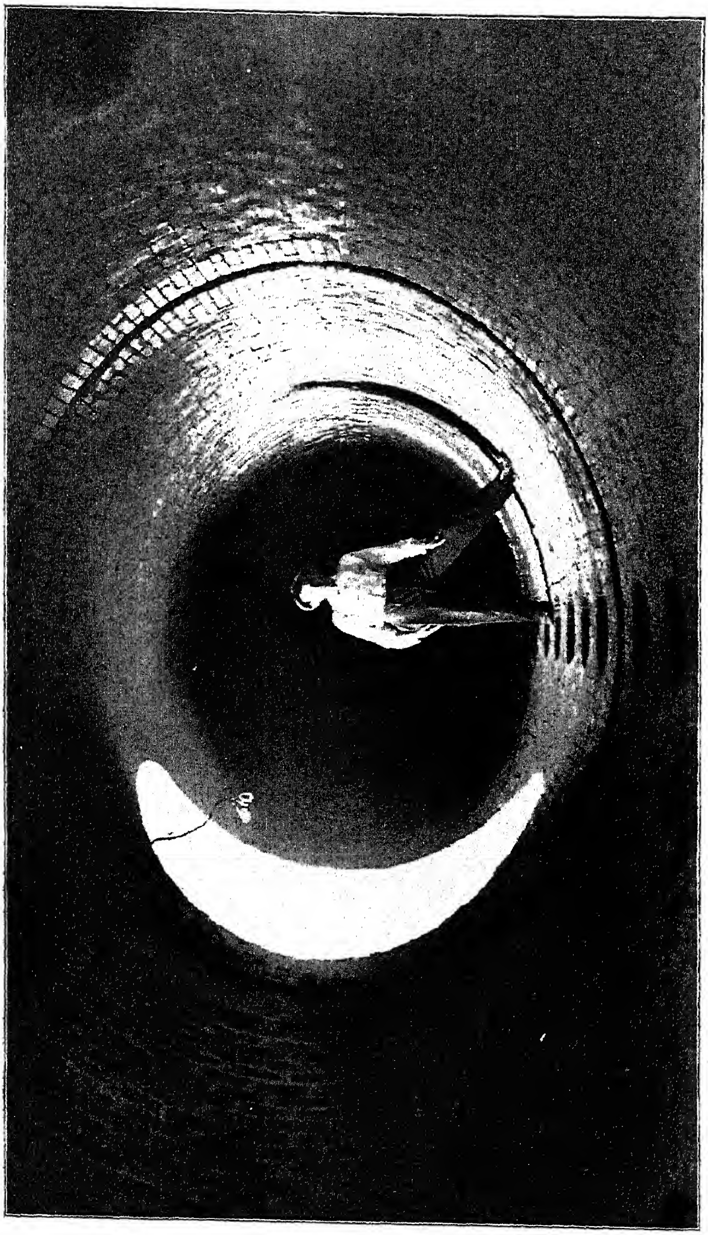


FIG. 7.—NINE-FOOT FLUE FROM FURNACE-TOP TO DUST-CHAMBER, SHOWING CONNECTION TO BY-PASS FLUE, AND SLOT FOR DAMPER, WHICH SERVES AS AN EXPANSION-JOINT.

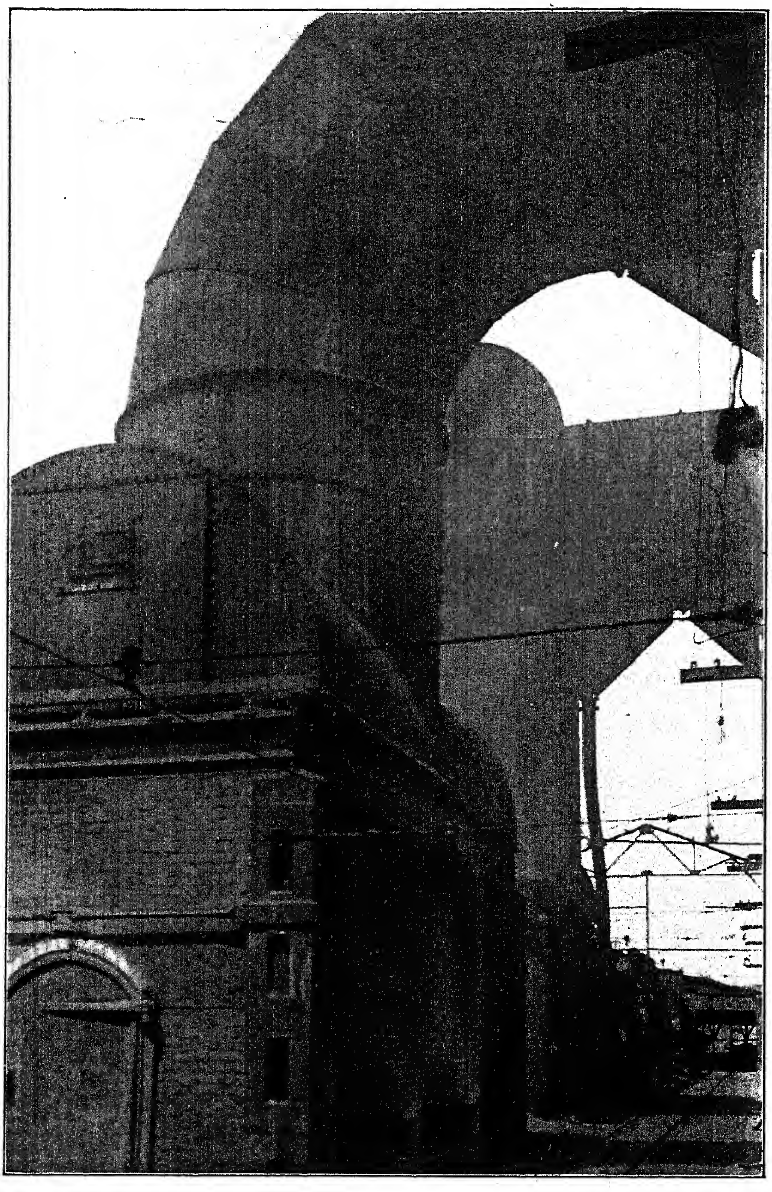


FIG. 8.—THE BRICK-LINED FLUES FROM NOS. 4 AND 5; BOTH FURNACES IN BLAST AND UNDER BACK PRESSURE.

available area to such an extent that the furnace would emit clouds of smoke through the doors, making work on the feed-floor almost unbearable. Moreover, accretions would form on the top and drop off, seriously disturbing the running of the furnace. This type of top would give satisfactory results if there were at all times a good draft on the flue.

Fig. 3 shows a thin sheet-iron hood, built to draw off the

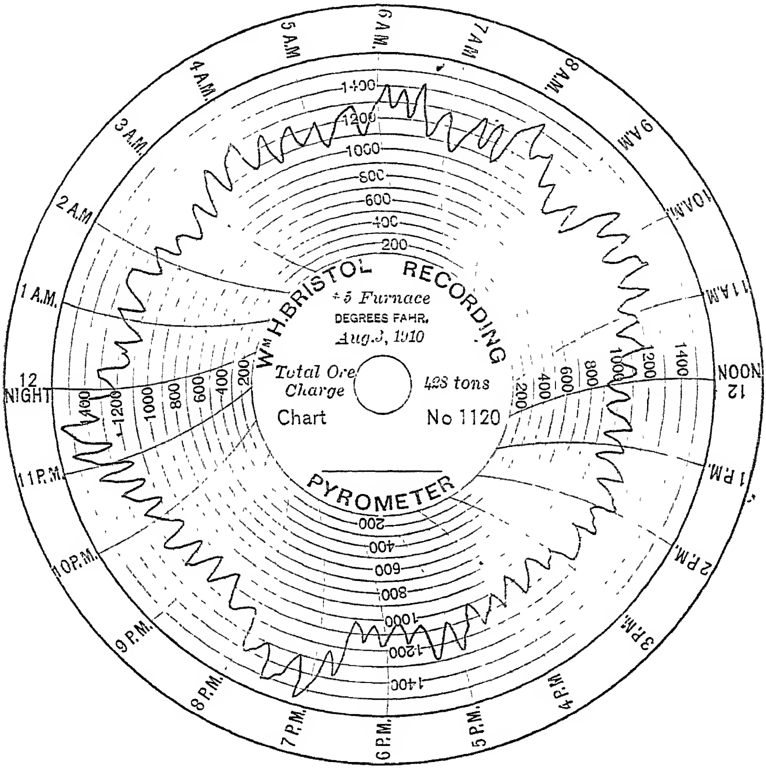
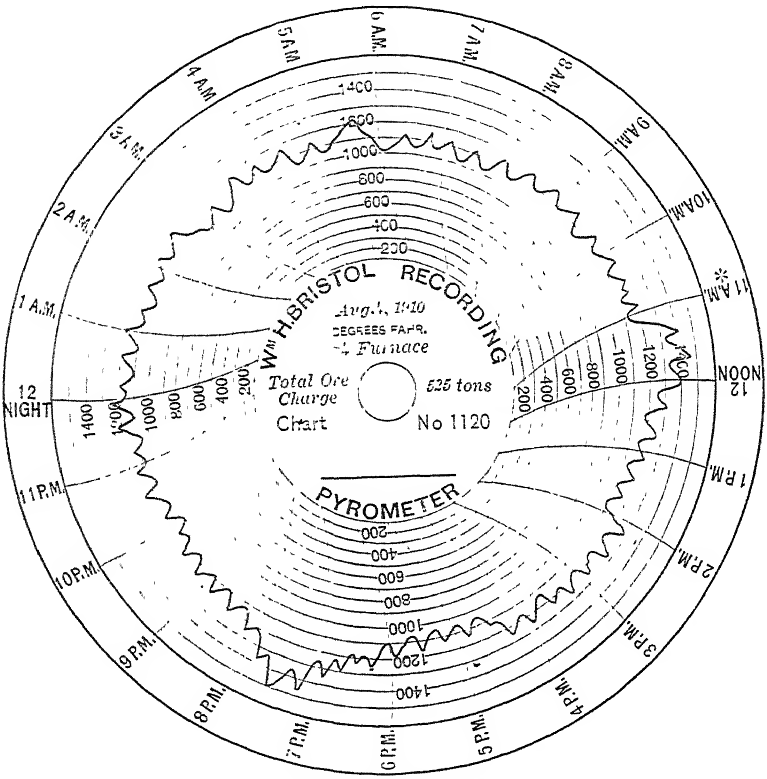


FIG. 9.—RECORD OF BRISTOL RECORDING PYROMETER, TAKEN AUG. 3, 1910, OF THE TEMPERATURES IN THE HORIZONTAL FLUE FROM NO. 5 FURNACE, WHEN SMELTING 428 TONS OF SULPHIDE ORE IN 24 HR. THE AVERAGE TEMPERATURE, BASED ON HIGH AND LOW POINTS, WAS 1,172 ° F., WHILE THE AVERAGE DROP IN TEMPERATURE, DUE TO CHARGING, WAS 202 ° F.

gas escaping from the doors when the furnace was working under a pressure. This would not be needed under ordinary furnace-conditions.

The latest developments in furnace-top construction at the Copperhill plant of the Tennessee Copper Co. are shown in

Figs. 5 and 6. Furnace No. 5, Fig. 5, was designed somewhat after the Shelby type (or "tin top," as it has been called), except that the top is made of cast-iron throughout from the feed-floor to the first brick arch. It is called the tubular-top furnace, and the flues are lined with tongue-and-groove brick clear through to the dust-chamber. The area of the flues totals 64 sq. ft., being more than twice the flue-area of No. 6



* Quartz decreased 11 a. m.

Fig. 10.—Record Taken Aug. 4, 1910, of the Temperature in the Horizontal Flue from No. 4 Furnace, when Smelting 525 Tons of Sulphide Ore. The Average Temperature for 24 Hr., Based on High and Low Points, was 1,163° F., while the Average Drop in Temperature, Due to Charging, was 100° F.

or No. 7 furnace, which had 31 sq. ft. each. No. 5 furnace went into blast for the first time June 19, 1910, and so far no cracks or troubles of any kind have developed. The doors of No. 5 are operated by air and open inward. This arrangement

is satisfactory so long as there is a suction in the furnace at the feed-floor level. It is an expensive top to build, and it is feared it will be expensive to maintain.

No. 4 furnace, shown in Fig. 6, is very simple. It has cast-iron corner-posts and dividers, with the walls and ends laid up of fire-brick. The 13-in. wall is unsupported by tie-rods or angles of any description. The half-circle top is supported on 20-in. I-beams resting on the columns, and all metal parts, as far as possible, are exposed to the cooling of the outside air. The large elbow and flue to the dust-chamber are 9 ft. in diameter inside the brick lining. The flues of both No. 4 and No. 5 are made of ¼-in. boiler-plate with 1.5 in. of magnesia blocks between the fire-brick lining and the boiler-plate. The doors of No. 4 open inward and are operated by a simple railroad two-throw weighted switch-stand. The door can be easily adjusted and repaired, and is quite tight, as it engages at the top and on the receiving-plate when closed.

Fig. 7 shows the flue from the tubular top of No. 5 to the dust-chamber. Fig. 8 is a view of the flues from No. 4 and No. 5.

The chart, Fig. 9, was taken by a Bristol recording pyrometer, with the fire-end in the middle of the flue from No. 5 furnace, about 35 ft. before it enters the dust-chamber, and shows the effect of the cast-iron top in the sharpness of the points or rapid drop in temperature at the time of charging. Fig. 10, taken from the flue of No. 4, likewise about 35 ft. from its entrance to the dust-chamber, shows the smaller variation in the temperature due to the fire-brick top. The average temperatures, based on the high and low points for the two furnaces, with the average drops in temperature due to charging, are shown in Table I.

TABLE I.—*Temperatures.*

Date.	Ore Smelted in Furnaces.		Furnace Recorded.	Average Temperature Shown by Chart.	Drop on Charging.	Glover Towers.	
	No. 4.	No. 5.				Temp.	SO_2.
	Tons.	Tons.	No.	Deg. F.	Deg. F.	Deg. F.	Per Ct.
Aug. 3..........	482	428[a]	5	1,172	202.3	1,164	4.87
Aug. 4..........	525[b]	505	4	1,163	100.9	1,205	5.45

[a] Ore-charge from which temperature was recorded. (Fig. 9.)
[b] Ore-charge from which temperature was recorded. (Fig. 10.)

The Glover temperatures are averages from half-hourly readings. The ore-charges of both furnaces are given in Table I., because the gases from both go into the large dust-chamber, where they mix, and after traveling approximately 300 ft. are drawn into the Glover tower, where the readings in the last two columns of the table were taken. These columns therefore represent both furnaces, whereas the two preceding columns represent, for Aug. 3, No. 5, and for Aug. 4, No. 4.

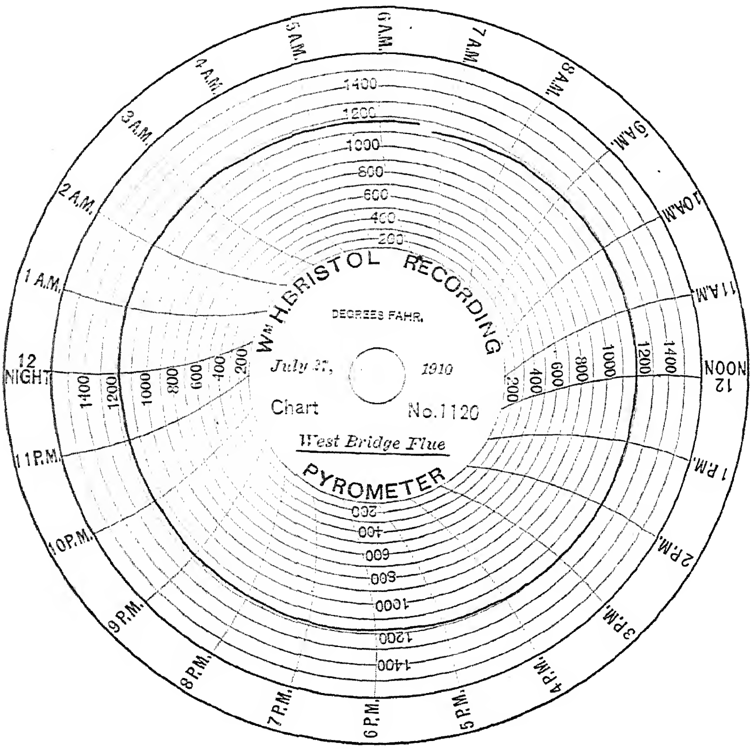


FIG. 11.—RECORD OF TEMPERATURE AT THE BASE OF THE GLOVER TOWERS. TAKEN JULY 27. NOTE THE STEADY TEMPERATURE, DUE TO THE EQUALIZING EFFECT OF THE DUST-CHAMBER, AS COMPARED WITH THE VARYING TEMPERATURES AT THE FURNACES, AS SHOWN BY FIGS. 9 AND 10.

An interesting feature of the comparison of the above figures is that the temperature in the horizontal flue of No. 5 on Aug. 3 was hotter than that in No. 4 on Aug. 4, when the latter smelted a much greater tonnage of ore. Another interesting

point is that, on the second day, the temperature is higher at the Glover tower than in the flue. Having but one recording-pyrometer, simultaneous readings were not obtained, but several days' observations indicate that the above results are a fair average. Now, there must be a loss in the heat of the gas between the furnace and the Glover tower, due to radiation; so the only reasonable supposition is that the furnace-gases contain some carbon monoxide or unconsumed sulphur, which mixes in the dust-chamber with oxygen from air-leakage or from the blast, and in burning increases the temperature acquired in the furnace-top. The oxygen-efficiency of the furnaces was probably from 75 to 80 per cent.

It has always been considered a fact at these works, that the sulphur is not completely burned in the furnace. Under old conditions, No. 6 and No. 7, when very hot, had a line of flame shooting out of the doors; and sublimed sulphur together with zinc oxide was deposited around the doors, at times, shortly after charging and when the furnaces were cold.

Fig. 11 shows the steady temperature maintained July 27, at the entrance to the Glover tower. The gases going to this tower came from No. 4 and No. 5, which smelted the following charges on that day:

Charges of July 27 (Fig. 11).

Materials.	No. 4 Furnace. Tons.	No. 5 Furnace. Tons.
Sulphide ore,	491	591
Quartz,	39	48
Total charge,	530	639
Coke,	25	28

These figures may be compared with the charges smelted on Aug. 3 and 4, Figs. 9 and 10, and with the irregular temperatures near the furnaces.

Charges of August 3 (Fig. 9).

Materials.	No. 4 Furnace. Tons.	No. 5 Furnace. Tons.
Sulphide ore,	482	428
Quartz,	48	42
Sintered flue-dust,	8	8
Charge,	538	478
Coke,	22	23

Charges of August 4 (Fig. 10).

Materials.	No. 4 Furnace. Tons	No. 5 Furnace. Tons.
Sulphide ore,	525	505
Quartz,	40	33
Total charge,	565	538
Coke,	23	26

For the month of September, the average gas (half-hourly tests) at the foot of the Glover tower showed 0.16 per cent. more SO_2 than the average of furnaces No. 4 and No. 5; and also a drop in temperature of 52° F. below the average of the furnaces recorded at the furnace-top.

The furnaces smelted the same average tonnage each (within 1 ton), per day, the average gas from No. 4 being 0.26 per cent. SO_2 less than the test at the Glover tower, and from No. 5 being 0.03 per cent. SO_2 below that at the Glover tower. The temperature from No. 4 averaged 22° F. higher than the gas at the Glover tower, and that in No. 5 averaged 81° F. higher than the gas at the Glover tower, indicating that on account of the larger tuyere-area and narrower furnace, a more complete combustion had taken place in No. 5 than in No. 4 furnace.

Some dimensions of the furnaces mentioned are:

	No. 4.	No. 5.	No. 6.	No. 7.
Length, inside jackets,	270 in.	270 in.	270 in.	270 in.
Width at tuyeres,	56 in.	48 in.	56 in.	56 in.
Height from tuyeres to feed-floor,	18 ft. 3 in.	18 ft. 3 in.	18 ft. 3 in.	18 ft. 3 in.
Tuyere-area per foot of furnace-length,	18.5 in.	49.5 in.	18.5 in.	49.5 in.

Recent Progress in Blast-Roasting.

BY PROF. H. O. HOFMAN, BOSTON, MASS.

(Canal Zone Meeting, November, 1910.)

I. Introduction.

The substance of this paper was prepared for the Seventh International Congress of Applied Chemistry, held in London, May, 1909, under the title, Some Developments in Blast-Roasting. In the absence of the author, a short abstract of the paper was read, and the abstract only will appear in the proceedings of the congress. The present paper has been partly rewritten and considerably enlarged by information gained since the original was prepared; also some illustrations have been added.

Blast-roasting[1] is a generic term for the process of forcing air through finely-divided metallic sulphide with the object of roasting and agglomerating in a single operation. At first the process was confined to a galena-concentrate; lime, limestone, or gypsum was added, to serve both as a diluent to keep separate the particles of galena that they might be thoroughly oxidized, and as a flux that the partly-roasted ore might be agglomerated by the formation of some slag. Later it was extended to treat other metallic sulphides.

The leading processes were the Huntington-Heberlein and the Savelsberg of Europe, and the Carmichael-Bradford of Australia. In the United States and Canada, the Huntington-Heberlein process has found application in some instances with galena-concentrates, for which it was originally intended; but this class of ore is not often treated alone, hence the process has had to be modified to suit conditions. The Savelsberg process has been transplanted to this country in about its original form. The Carmichael-Bradford is not used at present. The changes called for in the Huntington-Heberlein process to adapt it to the class of lead-bearing ores usually smelted have been the

[1] A. S. Dwight, *Engineering and Mining Journal*, vol. lxxxv., No. 13, p. 649 (Mar. 28, 1908).

cause of extending the operation of blast-roasting to other sulphides, especially to concentrates of sulphide copper-ores and to fines of sulphide copper-nickel ores, to prepare them for blast-furnace treatment. Thus an entirely new field has been opened up for the new manner of eliminating sulphur and sintering the roasted product. With it, methods and apparatus have come into use for obtaining a desulphurized porous clinker which are improvements upon those used originally in the three processes.

It is convenient to distinguish between the up-draft and the down-draft apparatus. Of the former, the various forms of the Huntington-Heberlein pot are typical; of the latter, the Dwight-Lloyd roasting-machine is the sole representative found at present in smelting-plants. The object of the present paper is to pass in review the present practice of blast-roasting in the United States and Canada by citing a few characteristic examples with the leading technical facts; the data have been obtained in part from the technical literature, but mostly, however, from the representatives of the works.

II. Up-Draft Blast-Roasting Apparatus.

1. *The Huntington-Heberlein Process.*—This process, in a form approaching the original, is in operation at three plants in British Columbia.[2] The works of the Consolidated Mining & Smelting Co. of Canada, Ltd., at Trail,[3] are the most important. Here the Huntington-Heberlein plant was installed in 1906 and enlarged in 1907–08; it consists of eight Huntington-Heberlein mechanical roasting-furnaces and 24 converting-pots. A roasting-furnace has a hearth 26 ft. in diameter, which makes 1 revolution in 3 min., and requires from 2⅓ to 3 h-p. to run it. There is one fire-box, 6 by 3 ft. in area. The angle of the blades in the stationary rake, and the rate of feeding, are so regulated that the thickness of ore on the hearth is from 4 to 5 in. The ore is a concentrate, which consists mainly of galena, but contains some pyrite and blende; an average assay shows:

[2] W. R. Ingalls, *Report of the Commission Appointed to Investigate the Zinc Resources of British Columbia. . . .* Mines Branch, Department of the Interior, Ottawa, p. 64 (1906). H. O. Hofman, *Mineral Industry*, vol. xv., p. 531 (1906.)

[3] Communication by J. Sabarthe, Manager. Paper by A. J. McNab, *Journal of the Canadian Mining Institute*, vol. xii., p. 424 (1909).

SiO_2, 18; Fe, 6; Pb, 56; S, 15 per cent.; some Cu; Ag, 65 oz. per ton. The charge is so made up that it will work well not only in the preliminary reverberatory-roast, but also in the blast-roast and in the smelting in the blast-furnace. Under normal conditions, the reverberatory roaster-charge is compounded so as to contain: Pb, 40 to 44; Fe, 10 to 13; SiO_2, 8 to 11; CaO, 7 to 10; and Zn, less than 10 per cent. When roasted and converted, the mixture will give a fairly hard and dense clinker of a yellow to grayish-yellow color, and not too tough to be readily broken to the required size; it will also make a small amount of fines. Charges containing more than 45 per cent. of lead have not proved entirely satisfactory, since the percentage of residual sulphur is found to be too high; the cause for which appears to be that some galena fuses before it is roasted. However, it is believed that by using a mechanical mixer to bring the components of the charge into closer contact, a lead-content even higher than 45 per cent. will be permissible. The lowest percentage of lead in the roaster-charge tried so far has been 38; and this has proved to be more satisfactory than 42, since it permits a faster sintering and gives less trouble; it is believed that under the given condition, a charge with from 38 to 40 per cent. of lead will fill all requirements.

As to the relation of iron and silica, experience has shown that the percentage of iron (Fe) ought to be at least equal to that of the silica, and that the charge works more satisfactorily if the percentage of iron exceeds that of silica by 1 or 2 per cent. When the reverse condition obtains there is always trouble in the blast-furnace, even if the elimination of sulphur remain the same, since the tonnage is diminished, the slags become richer in lead, and the heat creeps up in the shaft.

The amount of lime in the charge is allowed to range from 7 to 10 per cent.; it usually varies from 8 to 9 per cent. With less than 7 per cent., the results have been unsatisfactory. However, if with less than 7 per cent. of lime the total of iron and lime is kept about the same and the lead is held at 42 per cent., it is believed that the mixture will be as satisfactory as the one usually made. One disadvantage of a low lime-content is that the blast-roasted material is extremely tough. No charges have been run that exceeded 10 per cent. of CaO.

In rough-roasting, the ore passes through the furnace in 1 hr. 45 min.: the furnace puts through in 24 hr. from 38 to 45 tons of charge with a fuel-consumption of 150 lb. of coal per ton of ore, and an attendance per shift of one-fourth of the time of a fireman and one-half of the time of a wheeler; the rough-roasted ore retains about 7 per cent. of sulphide sulphur and 1.5 per cent. of sulphate sulphur. Experience has shown that for a good desulphurization in the converting-pots the ore discharged from the reverberatory furnace should not contain more than 9 per cent. of sulphur.

The temperature must be carefully regulated; if too low, the elimination of sulphur decreases; if too high, the furnace will rapidly become incrusted. If roasted properly the ore has been changed into sintered globules, in which none of the original constituents are discernible.

The elimination of about one-half of the sulphur in this rough-roasting operation is large as compared to the small amounts oxidized when a galena-concentrate low in other sulphides is being treated. Thus Guillemain[4] has shown that in roasting a pure galena-concentrate with a siliceous gangue, lead sulphide is mainly converted into sulphate, and the main object of rough-roasting such a concentrate previous to treatment in a converting-pot is to reduce its calorific value by changing part of the sulphide sulphur to sulphate sulphur, and thus counteract any premature fusion of the charge.

The converting-pot is 8 ft. 8.5 in. in diameter and 4 ft. 2 in. deep. The grating is made up of four sectoral perforated cast-iron plates, which are bolted together; these sections are preferable to a single casting as they do not crack so easily, because a change in size, due to expansion and contraction, is taken up in the joints. Cast-steel as a material for the grating has not proved satisfactory, since it buckles when heated. A pot takes a charge of from 12 to 15 tons; an analysis of such a charge gave SiO_2, 10.5; Fe, 10.3; Pb, 42.0; Cu, 1.3; S, 8.5; H_2O, 8 per cent., and Ag, 55 oz. per ton. The water-content of the analysis is exceptionally high; usually it is nearer 5 per cent. The roasted ore is discharged from the furnace into the boot of an elevator, which delivers it through a spray of water into

[4] H. O. Hofman, *Mineral Industry*, vol. xiv., p. 404 (1905.)

a brick bin, from which it is conveyed in cars to the iron hoppers of the converting-pots. In operating a pot, some slabs of wood are placed in the converter with a shovelful of glowing coals, a gentle blast is turned on until the wood burns freely, and then the charge is dropped through the hopper in the roof. The volume of blast is now increased so as to show a pressure of from 6 to 8 oz.; it is then gradually reduced until, towards the end of the blow, it has decreased to 2 oz., when the fire will have reached the surface of the charge. A charge is blown in 8 hr.; 95 per cent. of the material is in lump-form, leaving only 5 per cent. to be re-treated. An average assay of the coarse product shows Pb and Cu, 44; S, 3 per cent., and Ag, 60 oz. per ton. Since the introduction of the Huntington-Heberlein process, the lead-content of the blast-furnace charge has been considerably increased above the former standard, reaching 40 per cent. of the weight of the charge (ore plus flux). A charge averages 85 per cent. of blast-roasted material, the rest being some oxide lead-ore, siliceous ore, limestone, and foul slag. A furnace, 45 by 160 in. at the tuyere-section, smelts daily 170 tons of ore (not charge), and produces from 60 to 70 tons of lead-bullion.

An interesting development of blast-roasting at this plant is the treatment of the regular lead blast-furnace matte, which contains Pb up to 25, and Cu, from 8 to 10 per cent. The matte is granulated while it is being tapped from the fore-hearth, rough-roasted in an O'Hara or a Godfrey mechanical reverberatory furnace, whereby the sulphur-content is reduced to 12 per cent., the roasted matte moistened and blown in a Huntington-Heberlein pot, which usually lowers the sulphur-content to 3 per cent., occasionally to 1 per cent., and furnishes a basic material to be smelted with siliceous ore for a matte with about 42 per cent. of copper. Low-grade copper-matte with Cu, 15; S, 27; and Fe, 56 per cent. is treated in a similar manner.

A cake of blast-roasted matte is usually tough and therefore hard to break. At one plant in the United States this difficulty is in part remedied by spreading a layer of lime when half of the matte-charge has been dumped into the pot, and then covering it with the other half. This layer forms a division-plane along which the blast-roasted matte separates when it is discharged from the pot.

In many instances, blast-roasting of crushed lead-matte has been a failure at the start. The question whether granules and angular grains behave differently in the process has not been satisfactorily answered so far. However, several plants using the Huntington-Heberlein pots have succeeded in obtaining satisfactory results with matte. It thus appears that the difficulties encountered at first have been overcome.

The American Smelting & Refining Co., which owns the patents of the Huntington-Heberlein process for the United States, has introduced this method of blast-roasting in all of its lead-smelting plants, but with modifications necessitated by local conditions.

In treating a galena-concentrate, the mode of operating is comparatively simple, as the presence of a large amount of oxidized lead makes the scorification of the gangue relatively easy. Thus at East Helena, Mont.,[5] the leading ore is a galena-concentrate from the Cœur d'Alène district, Idaho, which assays[6] from 47 to 55 per cent. of lead and forms an easy charge for the 12 converter-pots which are in operation at present. When, however, charges are to be treated in which the lead-content is not to exceed 20 per cent., the matter assumes a different aspect, as besides a correct chemical composition of the mixture, its physical character becomes of decided importance. Thus, as a rule, a considerable amount of systematic experimenting becomes necessary before a suitable charge is obtained; it also takes some time before the required experience has been gained to know how to handle a given charge. Further, in a custom plant which treats ores from different sections of the country, the character of the ores received often undergoes changes sufficiently great to affect the result of blast-roasting, the percentage of sulphur eliminated becoming too small, or the amount of fines produced, too large. Hence the working of the Huntington-Heberlein process becomes complicated and is not always attended by uniform results.

The practice at the different works of the American Smelting & Refining Co. is not the same. The Murray plant, near

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[5] *Engineering and Mining Journal*, vol. lxxxvii., No. 7, p. 350 (Feb. 13, 1909).

[6] J. P. Rowe, *Mining World*, vol. xxx., No. 10, p. 428 (Mar. 6, 1909).

Salt Lake City, Utah,[7] built in 1901, is the largest lead-smeltery of the company. The introduction of the Huntington-Heberlein process in 1905 has changed the character of the blast-furnace work to this extent, that a furnace, 48 by 168 in. at the tuyere-section, using from 12 to 15 per cent. of blast-roasted ore, puts through 200 tons of charge in 24 hr. instead of 160 tons, and that the charge contains a larger percentage of ore than formerly. The plant contains five Godfrey mechanical roasters, 26 ft. in diameter, which work on the same principle as the Huntington-Heberlein roasting-furnaces, noted above. A roaster puts through in 24 hr. about 30 tons of ore containing from 18 to 25 per cent. of sulphur and reduces the sulphur-content to from 8 to 12 per cent., with an outlay of about 130 lb. of coal per ton of ore. The 14 Huntington-Heberlein pots of standard size, 9 ft. in diameter, are placed in a single row and are served by a 30-ton overhead electric traveling-crane; there is room on the opposite side of the building for a second row of 14 pots. A pot receives a charge of about 9 tons, and treats it on the average in 12 hr. A charge is made up of raw ore (S, high; SiO_2, 40; FeO, 20 per cent.) and roasted ore (S, 8 to 12; SiO_2, 10; FeO, 20 per cent.) in proportions to keep the lead between 18 and 20, the zinc under 10, and the sulphur between 16 and 20 per cent. In charging, there is first given a layer of ashes to protect the grate, this is followed by about 1 ton of hot roasted ore containing about 8 per cent. of sulphur, then comes another ton of similar ore with about 12 per cent. of sulphur, and lastly 7 tons of moistened mixture of raw and roasted ore, the water added being sufficient in amount to cause the particles to cohere when squeezed in the hand. The blast-pressure reaches 25 oz. and the sulphur-content is reduced to 4 per cent. The top of a blown charge is always made up of imperfectly desulphurized powdery material, which goes into another pot beneath the moistened ore-mixture. A blown pot is raised by means of the overhead crane, tilted to pour off the dusty fines, transferred to the breaking-platform at one end of the building, inverted, and the solid cake dumped on to conical

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[7] W. R. Ingalls, *Engineering and Mining Journal*, vol. lxxxiv., No. 12, p. 527 (Sept. 21, 1907); No. 13, p. 575 (Sept. 28, 1907). R. B. Brinsmade, *Mines and Minerals*, vol. xxviii., No. 5, p. 216 (Dec., 1907). H. O. Hofman, *Mineral Industry*, vol xvi., p. 665 (1907).

castings placed on the floor, which is laid with closely-set rails; large pieces are raised by means of the crane and dropped again; smaller ones go direct to a 10- by 20-in. Blake crusher, which discharges on to a conveyor.

The usual Huntington-Heberlein pot is 9 ft. in diameter and 4 ft. 6 in. deep. It is a single casting, which has in the bottom an air-pipe 5 in. in diameter; the height of the sectoral grate is 15 in.: the conical holes in the four, six, or even eight sections which make up the grate are $\frac{3}{8}$ in. in diameter. There is a tendency to improve upon the established form and thus do away with imperfections discovered in practice. The modern form of pot is larger and shallower than the older; a pot is 11 ft. in diameter and 3 ft. deep to top of grate, and takes 15 tons of a 50-per cent. lead charge instead of 8 or 9 tons. It is cast in four flanged sections, which are bolted together, and the bowl thus formed is bolted to a very flat bottom, the joints being made tight by an asbestos-cord packing. The life of a kettle is greatly increased by using sectional casting, a pot cast singly often cracking in the most unexpected manner.

The manipulation of the pots at the Murray plant is a great improvement upon the older method of dumping by means of worm-gear or lever.

The charging arrangements have also been brought up to modern standards of mechanical handling of materials.

2. *The Savelsberg Process.*—This process is used by the St. Joseph Lead Co., Flat River, Mo.,[8] for desulphurizing and agglomerating a non-argentiferous galena-concentrate resulting from the dressing of ore which occurs in dolomitic limestone. There are in operation 18 pots, placed in two rows. A pot is 8.5 ft. in diameter and 4.5 ft. deep; it treats in from 10 to 12 hr. a charge of 10 tons of a mixture having the following composition: SiO_2, 13.12; FeO, 5.90; CaO, 6.60; MgO, 3.30; Pb, 47.40; Zn, 2.10; S, 11.20; and H_2O, 6.00 per cent. The mixing of the charge is done mechanically[9]; the charge is delivered by a conveyor-belt to the boot of an elevator, which discharges into a bin above the charging-floor, from which the pots are filled by hand. The blast-pressure in the converting-pot is 10 oz. at the start, it is raised to 20 oz. during the blow, and in-

[8] Communication by O. M. Bilharz, Consulting Engineer.

[9] *Engineering and Mining Journal*, vol. lxxxix., No. 13, p. 648 (Mar. 26, 1910).

creased to 25 oz. towards the end; the amount of air used is from 1,000 to 1,500 cu. ft. per min. The finished charge contains from 10 to 15 per cent. of fines, which have to be re-treated. The solid cake is dumped and broken by hand. An analysis of the broken material is: SiO_2, 17.20; FeO, 9.00; CaO, 7.00; MgO, 4.00; Pb, 44.20; Zn, 3.80; and S, 2.30 per cent.

3. *Other Processes.*—At the Tintic Smelter, near Salt Lake City, Utah,[10] F. G. Kelley has substituted for the hemispherical pot a tilting tray-shaped steel vessel, 8 ft. long by 4 ft. wide by 14 in. deep, which holds a grate with $\frac{3}{8}$-in. holes and ribs to form sections 12 by 14 in. There is an under-grate blast and a movable hood. The tray receives, as a primer, 500 lb. of rough-roasted sulphide ore, and as a charge 3,000 lb. of mixed sulphide ore; the whole is blown in about 4 hours.

At the plant of the United States Smelting, Refining & Mining Co., at Bingham Junction, Utah,[11] there were in operation several Robinson blast-roasters. These have been replaced [12] by 20 so-called "roasting-boxes," of which 19 serve for ore-treatment and one for the supply of the primer for starting the ore-charges. A roasting-box has a hearth, 6 ft. square and 3 ft. deep, the bottom of which is made up of cast-iron plates perforated by $\frac{3}{8}$-in. holes, the sides are of fire-brick, and the hopper-shaped roof is of sufficient capacity to hold a 6-ton charge. The hearth forms the top of the wind-box, through which air is forced from a blower. At the back of the furnace is a door, 12 by 18 in., for the admittance of an electrically-driven ram, which pushes the roasted and clinkered cake out of the front through a sliding door, the upper part of which has a slot, closed by a slide, to furnish access to the furnace for leveling the charge and for closing blow-holes. The ram, similar to the one ordinarily used in discharging coke from horizontal retort coking-ovens, stands on an electric car and travels on a track parallel with the battery of 19 blast-roasters. The ore-charge consists of 33 parts of concentrates, from 5 to 10 of flue-dust, and from

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[10] Communication by M. J. Connolly.

[11] W. R. Ingalls, *Engineering and Mining Journal*, vol. lxxxiv., No. 12, p. 527 (Sept. 21, 1907); No. 13, p. 575 (Sept. 28, 1907). H. O. Hofman, *Mineral Industry*, vol. xvi., p. 668 (1907).

[12] Private communication by G. W. Heintz, General Manager. Papers by C. T. Rice, *Mines and Methods*, vol. i., No. 1, p. 6 (Sept., 1909), and L. A. Palmer, *Mines and Minerals*, vol. xxx., No. 8, p. 496 (Mar., 1910).

62 to 57 of fine ore. The composition of the ore is given in Table I.

TABLE I.—*Composition of Ore, Tintic Smelter, near Salt Lake City.*

Kind.	SiO_2 Per Cent	Fe Per Cent	CaO. Per Cent.	S Per Cent.	Cu. Per Cent	Pb. Per Cent	Zn Per Cent
Richmond fines,	7.0	32.0	7.5	0.8	0.1	4.0	5.0
Centennial Eureka, "L" fines,	70.0	6.4	4.0	0.7	1.2	3.5	0.6
Centennial Eureka, "O" fines,	71.6	6.1	3.4	0.5	1.8	0.5	0.9
Lead-plant flue-dust,	20.0	17.0	6.0	4.5	1.0	19.0	3 0
Bag-house dust,	.....	.....	.....	8.0	.....	43.0	6.3
U. S. concentrates,	4.6	19.4	2.0	33.7	1.0	19.1	4.8
U. S. concentrates,	4.0	23.3	2.0	34.2	0.8	17.9	11.6
U. S. concentrates,	5.2	22.2	2.0	34.0	1.0	20.6	14.6
U. S. concentrates,	4.4	25.2	2.0	33.5	1.0	18.9	9.7

The mixture is calculated to contain: S. 19; SiO_2, 28: Fe, 18; Pb, 13; and Zn, 6.5 per cent. The limit of coarseness of the individual particle is 0.5 in. The different ores are transferred to a Smith concrete-mixer and about 10 per cent. of water is added. The moistened mixture is then transferred to a bin by means of an elevator and thence charged into the hoppers of the roasting-boxes. The primer-mixture is made up of one part impure-blende concentrate, one part bituminous coal, and one and one-half parts of coke-screenings; the concentrate assays: Zn, 30.4: Pb, 6.8; Cu, 1.6; Fe, 12.7; S, 31.7; SiO_2, 7.6 per cent. (Au, 0.06, and Ag, 4.2 oz. per ton). The quantity necessary for an ore-charge is placed on the hearth of the 20th roasting-box, which is kept in operation more or less continuously, brought to a red heat by turning on the under-grate blast, and then transferred to the ore-roaster. At first, however, the grate is covered with a layer of limestone or siliceous ore to a thickness of about 2 in., then follows the red-hot primer, making a cover about 1.5 in. thick, and on top of this comes the charge of 6 tons of ore, which when leveled makes a bed about 26 in. thick. The doors of the furnace are now luted and blast is turned on to give a pressure of 2 oz.; the pressure is gradually increased until the end of the blow, when it reaches 9 oz., the blow lasting from 5 to 8 hr. During the working of a charge, the temperature is kept as low as possible in order to keep down the loss of lead and silver; the progress

of the blast-roast is watched through the slide in the front door. When no more sulphur-fumes are given off, and the roast is finished, the sintered cake is pushed into a sheet-steel boat by means of the ram, and then sprayed with water to cool the clinker and to wash off the fines. The boat is brought by means of a traveling electric crane to a 24- by 36-in. Farrel crusher set to 6 in.; after crushing, the clinker is hauled to the stock-bins of the blast-furnace. The 19 roasting-boxes treat about 320 tons of charge per day; the elimination of sulphur from ore-charges ranges from 65 to 70 per cent.; from matte-charges the average expulsion is only 37 per cent.; matte is therefore roasted in reverberatory furnaces. The loss in lead and silver averages 4 per cent. The plant is served, per shift, by 1 boss, 1 ram-man, 5 pot-men, and 2 men charging the ore to the bins and hoppers, or 9 men in all. The cost of treatment is $1.25 per ton of ore.

4. *Blast-Roasting of Sulphide Copper- and Copper-Nickel Ores in Pots.*—The changes in the Huntington-Heberlein process for treating lead-ores that were made necessary by the great variety of ores that have to be treated in a custom plant, led to the discovery that lime, although very desirable, as it increases capacity and gives a better product, was not necessary for the successful blast-roasting of lead-ores, and that it could be omitted entirely provided that the other components of the charge were correctly apportioned. This was the cause of W. R. Ingalls's term of "lime-roasting"[13] being in part replaced by L. S. Austin's designation of "pot-roasting."[14] The next step was to experiment with sulphide copper- and copper-nickel ores. In some cases the work has been successful, in others not. The main cause of this difference in results appears to lie in the physical character of the ore-charge. Slight changes in the chemical composition and the size of the ore-particles, which would hardly be noticed in a lead-charge, make the difference between success and failure in a copper-mixture. Fine concentrates, even when diluted with coarse ores, are liable to cause trouble. Copper-concentrates with an admixture of about 10 per cent. of flue-dust appear to work better

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[13] *Engineering and Mining Journal*, vol. lxxx., No. 10, p. 402 (Sept. 2, 1905).

[14] *Mining and Scientific Press*, vol. xciii., No. 17, p. 511 (Oct. 27, 1906).

than concentrates alone; with lead-mixtures the reverse has been found to be the case. Siliceous coarse ores which, when crushed to suitable size and mixed with the necessary bases, have resisted treatment, have been worked satisfactorily by crushing to about 1 in., blowing in a pot, crushing the caked and partly-desulphurized material to the usual degree of fineness, and then treating in the customary manner. Thus all

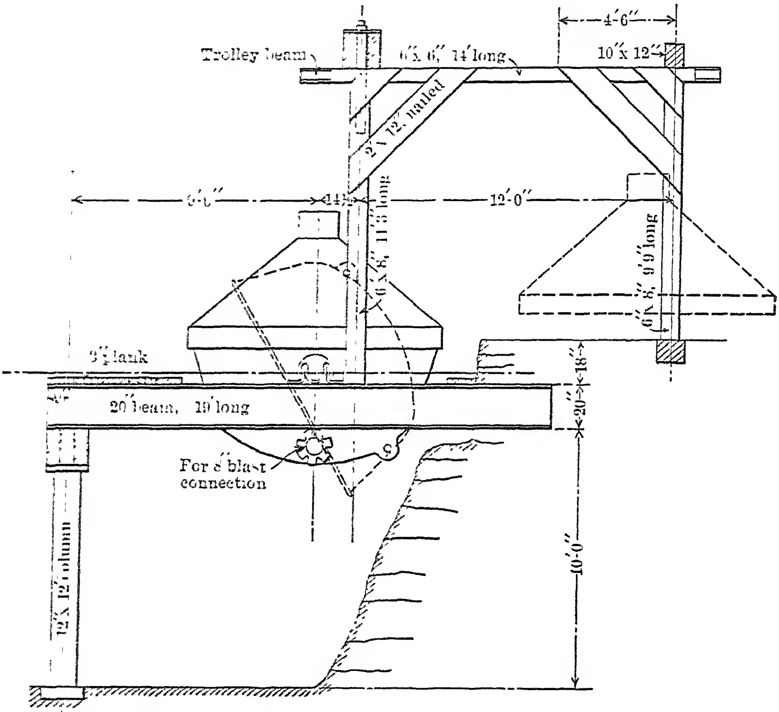

FIG. 1.—SIDE VIEW OF EXPERIMENTAL BLAST-ROASTING APPARATUS OF THE DETROIT COPPER MINING CO., SHOWING METHOD OF OPERATING.

sorts of twists and turns have had to be made in reaching out for success.

(*a*) The Garfield plant of the American Smelting & Refining Co., near Salt Lake City, Utah.[15] Here the work has so far not been entirely satisfactory. The raw material is a mixture of concentrates from different Utah copper companies, which contains from 25 to 35 per cent. of sulphur and about 8 per cent. of copper. It is rough-roasted in McDougall furnaces to reduce

[15] Communication by M. J. Connolly.

the sulphur-content to about 17 per cent. There are 25 converting-pots of the standard Huntington-Heberlein form; a pot treats 8 tons in 8 hr. The kindling-charge consists of 1 ton of hot calcines; the remaining 7 tons are cold calcines containing about 5 per cent. of water, and assaying: Cu, 8, and S, 17 per cent. The blast-pressure at the start is 6 oz.; it is increased during the run to 25 oz. and diminished towards the end to 20

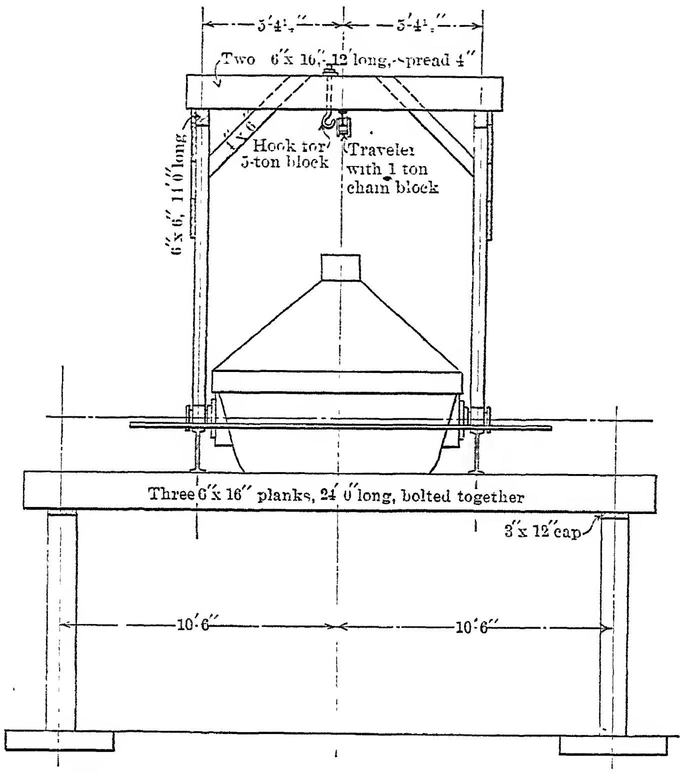

FIG. 2.—FRONT VIEW OF EXPERIMENTAL BLAST-ROASTING APPARATUS OF THE DETROIT COPPER MINING CO.

oz. The sulphur-content is reduced to 6 per cent., but from 40 to 50 per cent. of the weight of the charge has retained its original finely-divided character. Taking into consideration that the best metallurgical talent is at the disposal of this company, the difficulties with this class of ore in a Huntington-Heberlein pot appear to be very great.

(*b*) The Detroit Copper Mining Co. of Arizona, Morenci,

Ariz.[16] At these works sulphide copper-ore and flue-dust were blast-roasted for some time in spherical kettles. While the work was satisfactory from a purely metallurgical point of view, economic considerations were the cause of giving up the process. Some of the details, however, have permanent interest. The general arrangement of the plant is shown in Figs. 1 and 2; details of construction of the pot are given in Fig. 3. In Fig. 1 the pot is seen to rest with its trunnions on supports which are carried by two steel beams; these rest with one end on the solid rock and with the other on three planks bolted together, and they in their turn are supported by two columns. During a blast, the pot is held in position by four supports and a bearing-ring, Fig. 3. When a charge is finished, the hood is lifted with a 1-ton chain-block and conveyed to the position indicated by the dotted lines in Fig. 1, then the pot is tilted by means of a 5-ton differential chain-block, the cake falls to the ground, and is there broken by hand and shoveled into cars. The details of the construction of the pot and the grate are evident from Fig. 3.

The charge treated in a pot weighed 10 tons; it was made up of 8 tons of concentrate and 2 tons of flue-dust. A screen-analysis of the concentrate gave the following:

	Per Cent.
On 0.5-in. opening,	3.3
Through 0.5-in. and on 10-mesh,	9 6
Through 10-mesh and on 40-mesh,	30 9
Through 40-mesh and on 80-mesh,	25.5
Through 80-mesh and on 120-mesh,	14.0
Through 120-mesh and on 200-mesh,	7.8
Through 200-mesh,	8.9
	100.0

A chemical analysis gave the subjoined constituents: SiO_2, 14.3; Fe, 24.9; Al_2O_3, 5.0; CaO, 0.5; MgO, 0.5; S, 32.2; Cu, 18.8; difference, 3.8 per cent. The moisture in the concentrate as used averaged 8.5 per cent. Flue-dust gave upon analysis: SiO_2, 23.9; Fe, 25.9; Al_2O_3, 5.5; CaO, 1.9; MgO, 0.5; S, 16.2; Cu, 17.1; difference, 9.0 per cent. As the plant has no roasting-furnaces which would furnish hot material to serve as a primer, the usual procedure had to be changed. The grate was

[16] Communication by L. R. Wallace, Metallurgist.

covered with a layer of ashes 0.75 in. thick, a small fire of waste and wood was started in the center and then, when burning well,

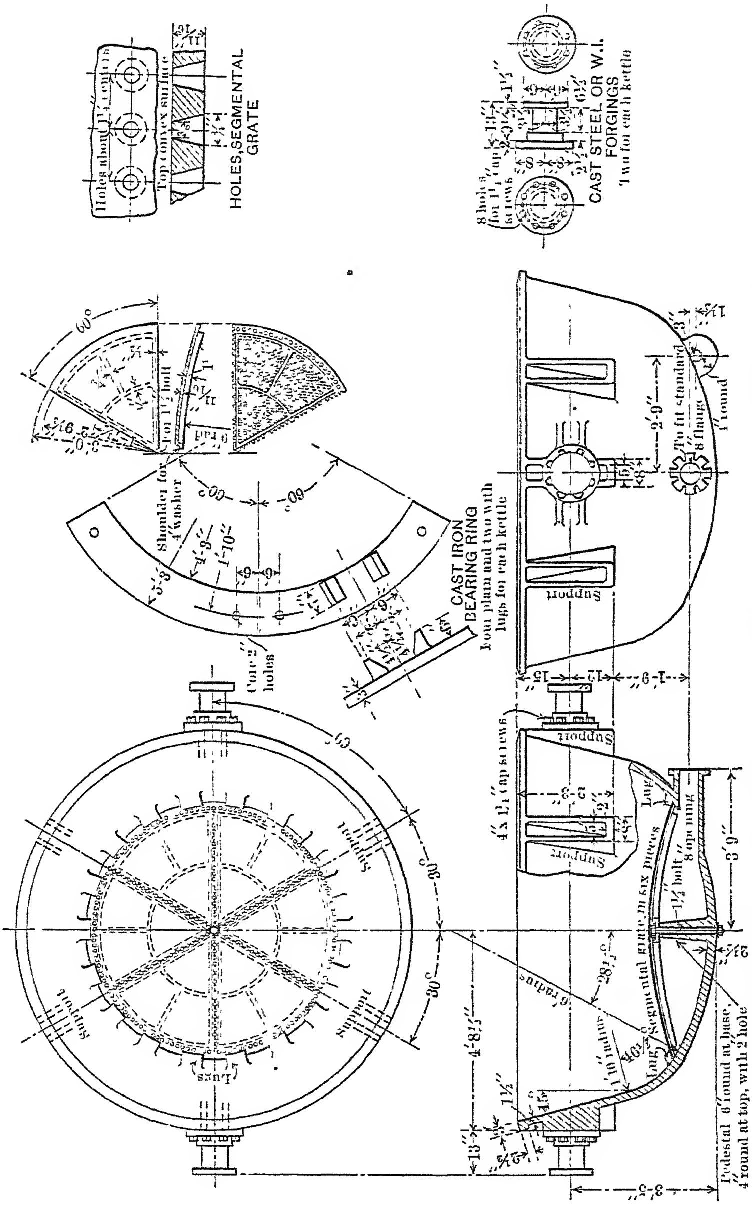

FIG. 3.—DETAILS OF CONSTRUCTION OF THE ROASTING-POT OF THE DETROIT COPPER MINING CO.

75 lb. of saw-dust was fed in such a way as to be about 6 in. deep at the center and to taper towards the periphery. Now a

charge of 2 tons of warm (80° C.) flue-dust was given, and this was followed by a 3-in. layer of concentrate; the blast was started with a pressure of about 2 oz. and allowed to act for 30 min., which was sufficient to kindle the concentrate thoroughly. Next the air-valve was opened full, giving a pressure of about 18 oz., and the concentrate was fed in as fast as the heat crept up. A charge of 2 tons of flue-dust and 10 tons of concentrate required 20 hr. for treatment: one of equal parts of flue-dust and concentrate, 14 hr. The best results gave 93 per cent. of coarse material, which upon analysis showed: SiO_2, 17.6; Fe, 34.5; Al_2O_3, 5.8; CaO, 0.8; MgO, 0.6; S, 9.5; Cu, 21.4; difference, 9.8 per cent.

(*c*) The Canadian Copper Co., Copper Cliff, Ont.[17] At these works copper- and nickel-bearing pyrrhotite is roasted in heaps, the roasted ore is smelted in blast-furnaces, and the resulting matte brought forward in barrel-converters to 80 per cent. of nickel-copper. There are no roasting-furnaces to treat the fines made in breaking the ore, which are usually charged raw into the blast-furnace with the heap-roasted ore. Experiments with pot-roasting were successful, but the process was not carried further, as in the roast-yard there were no mechanical appliances, and working by hand made the cost of treatment too high. The pot used was 8 ft. 6 in. in diameter and 6 ft. deep; the arched grate, with $\frac{5}{8}$-in. holes, was 2 ft. above the center of the kettle. Two classes of ore, both 0.5 in. and smaller, were treated, viz., Cream Hill fines, and a mixture of 4 volumes of Creighton ore with 1 volume of flue-dust. A pot received a charge of 5.75 tons of Cream Hill fines, which assayed SiO_2, 31; Fe, 19; CuNi, 6.15; S, 10; H_2O, about 1; while 6 tons of mixture, also with about 1 per cent. of water, could be worked in a single operation. The Creighton ore assayed: SiO_2, 15.5; Fe, 42; CuNi, 6.6; and S, 26 per cent.; the flue-dust, SiO_2, 23; Fe, 7.3; CuNi, 6.8; and S, 8 per cent. The mode of operating in both cases was the same. Kindling is distributed over the grate to cover an area 2 ft. 6 in. in diameter; on top of this 2 pails of coke-breeze is spread in a layer about 1 ft. in diameter; the wood is ignited, half-blast turned on until the coke is well kindled, ore-charging begun, and then full blast is turned on.

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[17] Communication by D. H. Browne, Metallurgist.

When the ore begins to burn in the center, the coke is worked towards the periphery to draw outward the fire and thus ignite the ore outside of the central area. When a good layer of ore has been well kindled, the pot is filled with the rest of the charge. About 7,500 cu. ft. of air is blown in per minute at a pressure of 15 oz.; the time of working is 8.30 hr. The pot-roasted Cream Hill ore retained 2.75 per cent. of sulphur and gave from 15 to 20 per cent. of fines; the Creighton mixture of ore and flue-dust, 12 per cent. of sulphur and from 20 to 25 per cent. of fines.

III. Down-Draft Blast-Roasting Apparatus.

Blast-roasting with up-draft pots is accompanied by several disadvantages. The leading ones are: (1) the ore is exposed for a long time to the influence of hot gases, and this with lead-ore means loss of metal; (2) the process is intermittent; (3) the pot requires more or less constant attention in filling, blowing, and discharging; (4) the amount of fines that has to be re-treated is liable to be large; (5) the handling of fines causes much loss by dusting; (6) the cake is unevenly sintered; (7) the breaking of the cake is expensive, even when done mechanically, and it is unhealthy if done by hand. Thus the cost of the Huntington-Heberlein process in treating lead-ores is calculated by Ingalls[18] at $2.16 per net ton of blown mixture, or $2.70 per net ton of ore. This figure is somewhat high; deducting for blown mixture one-third for the Rocky Mountain plants, and one-half for the Mississippi Valley region, comes nearer to the actual facts. These disadvantages appear to have been overcome by the Dwight-Lloyd sintering-process and apparatus, in which the ore is exposed only for a short time to the influence of heat and blast; that is, approximately 1 min. for each 1 per cent. of sulphur; the process is continuous, and when once adjusted requires little or no attention; it makes very little fines and furnishes a porous sinter of coke-like structure, which usually is small enough to go direct to the blast-furnace.

Three types[19] of sintering-machines are at present in operation: the drum-machine, the straight-line machine, and the horizontal-table machine. All embody the same three func-

[18] *Trans.*, xxxvii., 641 (1907).

[19] Communication by A. S. Dwight.

tions: (1) a layer of ore is spread mechanically to the thickness of from 2.5 to 5 in., average 4 in., on a traveling herring-bone grate; (2) the ore-stream thus formed passes first under an igniter, placed at right angles to the grate, to become ignited at the surface, and then over a stationary suction-box, which by down-draft causes the roasting started at the surface to progress downward and be finished when the ore reaches the farther end of the section-box: (3) the sintered ore is discharged automatically in a size suitable for blast-furnace treatment. A Dwight-Lloyd shallow tray[20] using down-draft, but working intermittently, is in operation at the works of the Cerro de Pasco Mining Co., Peru, for desulphurizing mixtures of sulphide copper-ore and flue-dust.

1. The drum-machine,[21] shown in Fig. 4, consists of a horizontal cylinder, 11 ft. 4 in. in diameter and 3 ft. face, made up of a pair of circular rims of iron carrying cast-iron herring-bone grates with an effective width of 30 in.; the drum-shaped structure rests on two pairs of friction-rollers, one of which is connected with the power. Inside the drum is a stationary suction-box, which occupies the top quadrant of the circle. The ore, fed mechanically upon the rising grate, is ignited, as shown in the illustration, by gasoline-jets, travels over the suction-box in about 20 min., and is discharged automatically from the grate-surface by the points of an upturned grizzly. One experimental machine at the works of the American Smelting & Refining Co., Maurer, N. J., treated, without addition of lime, 50 tons of galena-concentrate, with Pb, 50 per cent., at the rate of 30 tons in 24 hr., with a 4-in. layer of ore, a vacuum of 4 oz., and a power-consumption of 12 h-p. The sulphur-content was reduced to 3.4 per cent., and gases filtered through the bags showed the loss in metal to be less than 0.5 per cent.

This same machine is now installed at the works of the Baltimore Copper Smelting & Rolling Co., Baltimore, Md., for treating, at intervals, mixtures of sulphide copper-ore concentrates and flue-dust at the rate of 25 tons per 24 hours.

In general, it may be stated that the revolving drum consumes from 0.75 to 1.5 h-p.; the fan, from 10 to 20 h-p.; the

[20] *Mining and Scientific Press*, vol. xcviii., No. 5, p. 195 (Jan. 30, 1909).

[21] *Engineering and Mining Journal*, vol. lxxxv., No. 13, p. 649 (Mar. 28, 1908). *Mineral Industry*, vol. xvi., p. 380 (1907).

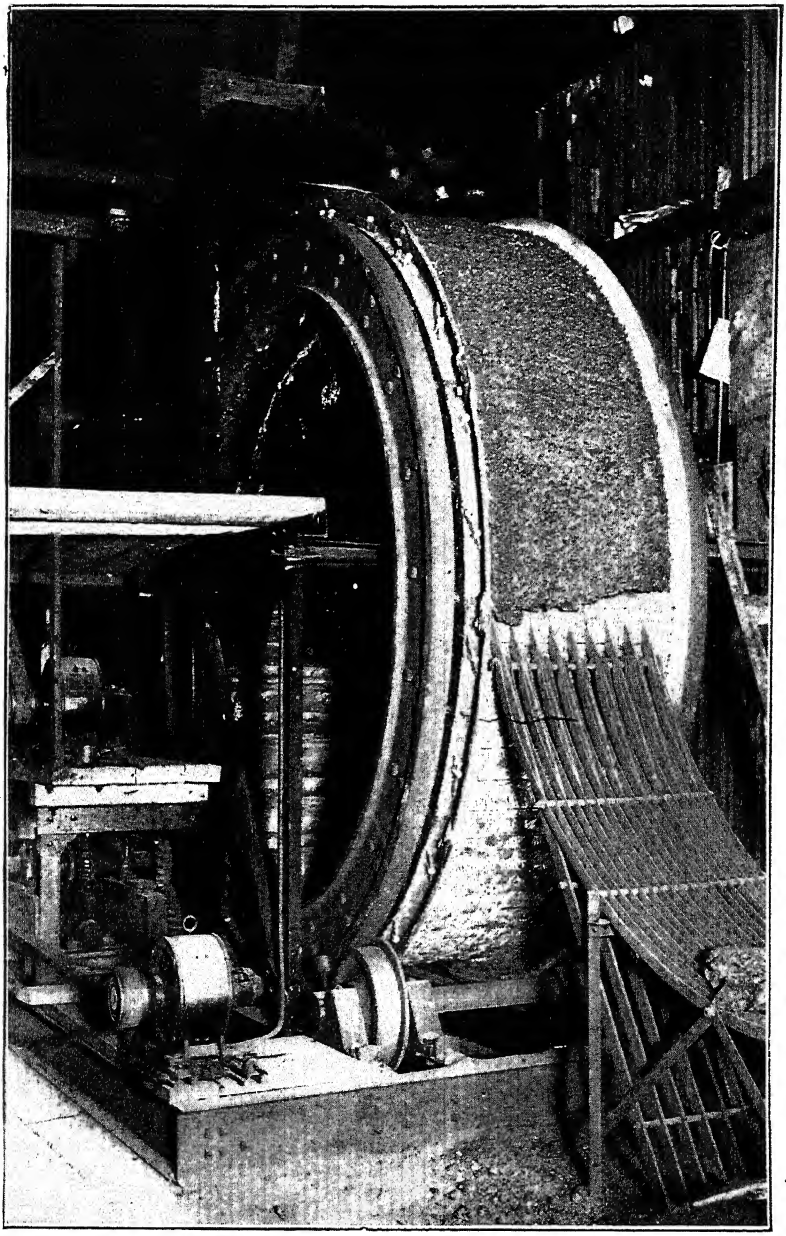

FIG. 4.—DRUM-FORM OF DWIGHT-LLOYD BLAST-ROASTING APPARATUS, SHOWING UPTURNED GRIZZLY FOR REMOVAL OF SINTERED PRODUCT.

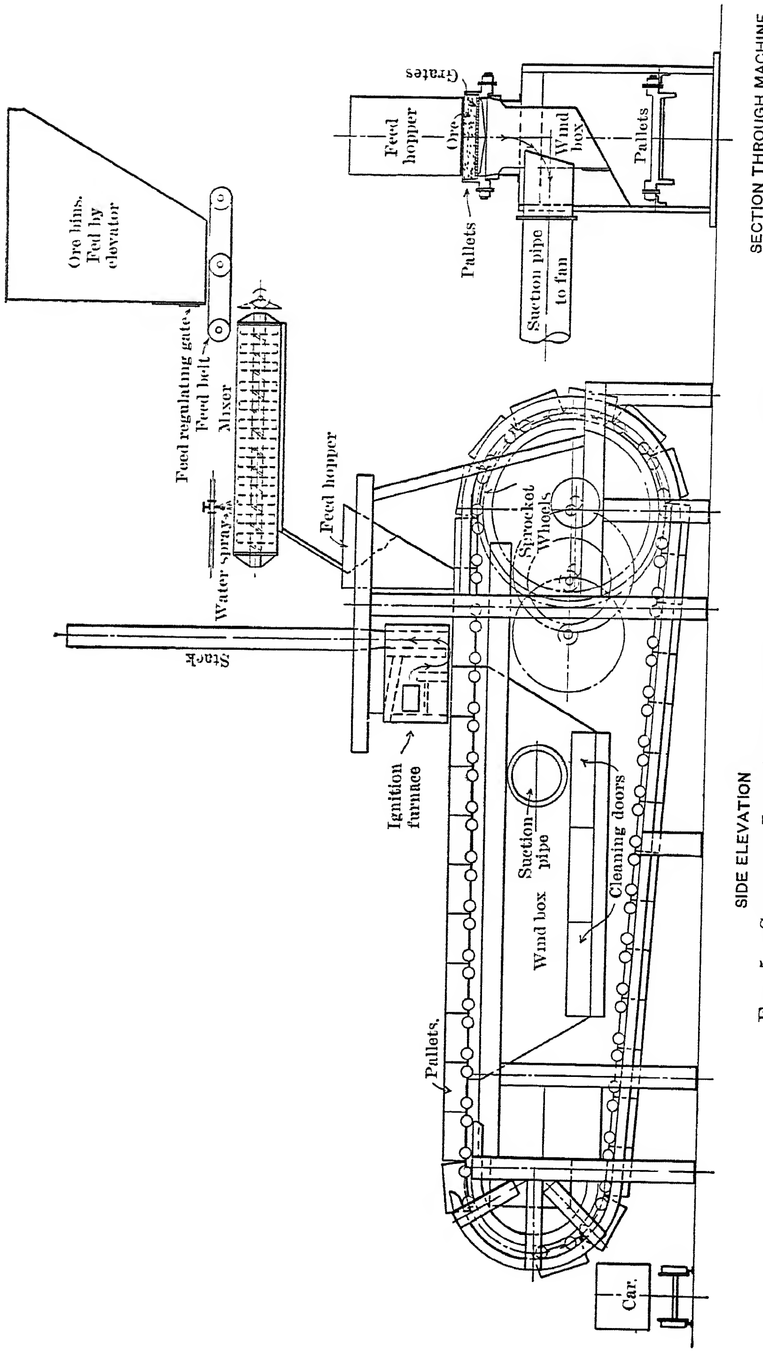

FIG. 5.—STRAIGHT-LINE DWIGHT-LLOYD BLAST-ROASTING MACHINE.

mixing, elevating and conveying apparatus, from 4 to 5 h-p., giving a total of from 15 to 25 horse-power.

2. The straight-line machine, shown in Fig. 5, consists of a frame of structural steel supporting a feeding-hopper, an igniting-furnace, a suction-box, and a pair of endless-track circuits to accommodate a train of small truck-like elements called pallets which, in combination, form practically an endless conveyor, with the continuity broken at one place in the circuit. Each pallet is provided with four wheels, which engage with the tracks or guides at all parts of the circuit, except when the pallet is passing over the suction-box, and then the pallet slides on its planed bottom over the planed top of the suction-box, thus making an air-tight joint. A pair of cast-steel sprocket-wheels, turning inside of concentric guide-rails, lift the train of pallets from the lower to the upper track by engaging their teeth with the roller-wheels, and launch each pallet in a horizontal path under the feed-hopper and igniting-furnace, and over the suction-box. In a train of pallets in action, all the joints are kept closed and air-tight, by the pallet being pushed from behind. At the beginning and the end of the track formed by the planed top of the suction-box, there is a planed "dead-plate" over which the pallets must glide; it serves to prevent any leakage of air. After a pallet passes over the suction-box and terminal dead-plate, its wheels engage the ends of the circular discharge-guides. These are adjusted with the view of raising the pallet about 0.5 in. vertically and thus automatically prying up the cake of sinter and freeing it from the grate-slots. A "breaking-roller" prevents the prying action from extending too far back, and tends to form a line of fracture. This roller, however, is not essential in all cases. On reaching the curve of the guides, the pallets one by one drop into the guides, each strikes the pallet which has preceded it and, at the same time, discharges its load of sinter-cake, and shakes free the slots of the grates. The force of the blow can be regulated by the gap left in the train of pallets at this point. The weight of the train keeps the pallets fed down to the lower teeth of the sprocket-wheels.

The igniter frequently used with this machine is a small coal-burning furnace built of tiles, having a grate-area of 10 by 30 in. and burning 500 lb. of coal in 24 hr. The flame after passing over the fire-bridge is deflected downward upon the ore by a

brick curtain that can be raised and lowered, and then is drawn upward by the natural draft of a small stack or bleeder.

The suction-box on top is 12 ft. 6 in. long and 30 in. wide, and gives for the grates an effective hearth-area of 31.25 sq. ft.; this is the true measure of the capacity of the machine. The pallets, 30 in. wide by 18 in. long, weigh with grates 550 lb.

The power delivered to the machine has its speed-factor reduced by passing through a train of gear-wheels, the last of which engages the internal gear-teeth cast in the large sprocket-wheels, and actuates the train of pallets.

The complete cycle of operations is as follows: A pallet, being pushed onward tangentially from the top of the sprocket-wheels, passes under the feed-hopper, where it takes its load in the form of a continuous even layer of charge, say 4 in. thick, passes next under the ignition-furnace, where the top surface is kindled, and at the same time comes within the influence of the downward-moving currents of air, induced by the suction-draft; these carry the sintering action progressively downward until it reaches the grates. The roast-sintering operation is complete, the cake is discharged by dropping into the discharge-guides, the pallet crowds its way back to the sprocket-wheels, is slowly raised to the upper tracks, and begins a new cycle.

A straight-line machine of the size described with effective area of 31.25 sq. ft. weighs, without accessories, about 16 tons.

At the works of the Ohio & Colorado Smelting & Refining Co., Salida, Colo.,[22] the machine in operation is 30 ft. long; the distance between the deflecting brick curtain of the igniter and the surface of the ore is 2 in.; the rate of travel of the pallets is 8 in. per min. The ore-mixture comprises ores of the company of the composition given in Table II.

TABLE II.—*Composition of Ore-Mixture, Ohio & Colorado Smelter, Salida, Colo.*

Kind.	Ag. Oz. Per Ton.	Au. Oz Per Ton.	Pb. Per Cent.	Cu. Per Cent.	Insol. Per Cent.	Fe. Per Cent.	Zn. Per Cent.	S. Per Cent.
New Monarch, . .	?	?	3.0	none	50	14	9	18
Flue-dust, ...	17.7	0.46	17.4	1.4	25.4	17.8	4 5	5.9
Sulphide concentrate,	?	?	23.5	none	6.2	19.8	10.2	28.6
Siliceous ore fines, .	25	3.0	5	none	65	8	?	6
Leadville oxide fines,	10	......	3	none	12	48	......	1

[22] Communication by M. J. Connolly.

A typical charge for the machine by weight is: New Monarch, 40; flue-dust, 15; sulphide concentrate, 30; siliceous and oxide ore, 15 per cent.; this brings the SiO_2 content to about 35 and the S to 17.7 per cent. In working, it was found that the largest permissible size of ore-particle was 0.25 in., and that the quantity should not exceed 25 per cent. of the charge. The sintered material retains 4 per cent. of sulphur. The machine treats 50 tons of mixture in 24 hr.: the suction in the chamber shows a vacuum of 6 oz.: the igniter consumes 1,000 lb. of coal per 24 hr.: 1 man attends the machine, but can look after several; the cost of treatment, including bringing the ore 200 ft. and moving sinter a similar distance, is $0.75 per ton.

Two machines of this type are in operation at a lead-smelting plant in Illinois, where the cost of treatment at the rate of 100 tons in 24 hr. is less than $0.50 per ton. Other machines are doing satisfactory work in the Mississippi valley. At East Helena, Mont., raw matte from the lead blast-furnace is successfully worked by the machine, the matte forming 62 per cent. of the weight of the charge.

In general, it may be said that a machine treating from 35 to 40 tons of copper-lead ores per day requires, for mixing of charge and propulsion of pallets, from 3 to 6; for the fan, from 12 to 20: or a total of from 15 to 26 h-p. When treating a galena-concentrate, averaging 50 per cent. of lead in the mixture, at the rate of 60 tons of charge per 24 hr., the power required has been found to be less; for mixing and propelling, from 2 to 3: for the fan, from 10 to 13: or a total of from 12 to 15 horsepower.

3. The horizontal-table machine resembles a horizontal rotating picking-table, in which the ring-shaped table is replaced by herring-bone grates. The outer diameter is 15 ft., the inner 8 ft., giving a total grate-area of 126 sq. ft., of which about 50 per cent. is effective. The table makes about 1 rev. in 45 min. There is a stationary feed-hopper and igniter; the sinter is removed by means of a scraper and a deflecting-apron. This type of machine has been tried under a variety of conditions, ranging from medium-grade lead-ore and flue-dust to fine copper sulphide concentrate, and has given good results. It is a convenient form for a large unit, but it has the great dis-

advantage that the scraper in removing the sinter forces an undue amount of fines through the grate-slots. Such a machine is in operation at the Garfield plant of the American Smelting & Refining Co., Utah, treating per day about 35 tons of a fine copper sulphide concentrate, of which 45 per cent. will pass a 200-mesh screen. The sulphur in the raw charge amounts to about 30, and in the sintered product to about 6 per cent. Other machines of this type are in operation in other plants of this company, but machines of the straight-line type have been chosen for the more recent installations.

The following data may supplement the details given with the three types of machines that are in operation at present:

For a grate-surface of 30 sq. ft., treating a charge with about 15 per cent. of sulphur, from 3,000 to 4,000 cu. ft. of hot gases have to be handled per minute. The vacuum varies from 2 to 7 and averages 5 oz. The wheel of the fan which produces the suction has to be cast heavier than ordinarily in order to withstand the wear of fine particles that pass through the grates. The Salida fan has a wheel 32 in. in diameter, 10-in. face, 13-in. inlet and outlet, runs from 1,500 to 1,900 rev. per min., and consumes from 10 to 20 h-p. The temperature of the gases at the fan ranges from 100° to 150° C. The latest practice is to use large, slow-speed fans. Such a fan has an impeller wheel 66 in. in diameter, with a 7-in. face, and makes 660 rev. per min., and handles about 3,600 cu. ft. of gas at a temperature of 180° C., creating a vacuum of 4 oz. and consuming 19 i.h-p. This volume corresponds to a machine having a suction-box 30 by 150 in. Running the fan at 800 rev. per min. makes it answer for two machines.

In treating pyritic ore some sulphur in driven off and vapor deposited in the casing, whence it is removed at intervals through clearing-doors. The amount of fines which falls through the grate has been found to range from 3 to 5 per cent. of the weight of the charge; the bulk of this material is removed through doors in the suction-box.

In making up a charge, it is essential that the constituents be intimately mixed and uniformly moistened; the amount of water added varies with the character of the ore and ranges from 6 to 10 per cent.

As to the chemical composition, the charge-components

ought to be so apportioned as to furnish a slag that forms at a low temperature and requires little superheating to become fluid. The range of SiO_2 lies between 10 and 35 per cent.; it is better to have FeO in excess of CaO than *vice versâ ;* Pb can be high or low; charges with Pb. 60 per cent., are being run successfully; S, as low as 10, but a good average is 18 per cent., while with more than 20 per cent. the process is much retarded. However, in treating copper-ores with which the sulphur-content has to be reduced only to 8 or 10 per cent., the charge can contain as much as 25 per cent. of sulphur.

The Reduction of Calcium Sulphate by Carbon Monoxide and Carbon, and the Oxidation of Calcium Sulphide.

BY H. O. HOFMAN,* BOSTON, MASS., AND W. MOSTOWITSCH,† TOMSK, SIBERIA.

(Canal Zone Meeting, November, 1910.)

I. Introduction.

In a previous paper,[1] The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes, we published the results of our investigation into the behavior of calcium sulphate in air; that is, under oxidizing conditions. The results as a whole hold good for a neutral atmosphere, or one which has no chemical effect upon calcium sulphate. The present paper deals with the behavior of calcium sulphate under reducing conditions, the reducing-agents employed being carbon monoxide and carbon, and the changes the calcium sulphide formed undergoes when it is subjected to an oxidizing-roast.

Technical literature deals very sparingly with the subject. The information furnished by the recent leading chemical manuals is general. Thus Dammer,[2] Moissan,[3] and Abegg[4] quote Berthier as having reduced, in 1823, calcium sulphate with carbon, and Stammer as having done the same in 1851

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* Professor of Metallurgy, Massachusetts Institute of Technology.
† Metallurgical Instructor, Imperial Technological Institute.

[1] *Trans.*, xxxix., 628 to 653 (1909).
[2] *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 309 (1894).
[3] *Traité de Chimie Minérale*, vol. iii., p. 543 (1904).
[4] *Handbuch der anorganischen Chemie*, vol. ii., part ii., p. 116 (1905).

with carbon monoxide: Gmelin-Kraut[5] add Leplay and Laurent (1848) as third investigators. Metallurgical authorities confine their statements to the known fact of reduction by carbon.

II. Preparation of Raw Materials.

1. *Calcium Sulphate.*—This salt was prepared in the same manner as described in our previous paper,[6] and was found to be chemically pure.

The results of the present research furnished a new method of testing the purity of sulphates, like calcium or barium sulphate, which are completely reduced without loss of sulphur to sulphides by carbon monoxide at temperatures lying 100° C. or more below those at which they are dissociated. It consists in reducing a given quantity of sulphate in a current of carbon monoxide to sulphide, and then comparing the weight of the sulphide formed with that theoretically required. Calcium sulphate[7] begins to be decomposed at 1,200° C. when heated in pure dry air, barium sulphate[8] at 1,500°. The operation consists simply in heating in an electric-resistance furnace calcium sulphate, placed in a porcelain boat, in a current of dry air at 1,000° to constant weight, and reducing the amount obtained at from 900° to 950° in a current of carbon monoxide until this reduction is complete, that is, constant weight has been reached. Two cases may serve as examples. A sample of $CaSO_4$ calcined at 1,000° weighed 0.3825 g.; heating for 1.5 hr. at from 910° to 920° in a current of CO gave 0.2022 g. of CaS; theory calls for 0.2027 g.; continuing the heating at from 950° to 960° for 1 hr. more caused no change in weight. Another sample of calcined $CaSO_4$ weighed 0.6336 g.; reduction at from 920° to 950° to constant weight gave 0.3357 g. of CaS; theory calls for 0.3358 g. This reduction method is more accurate than the analytical, in that no chemical reagents are used except CO, and this is easily obtained as a chemically-pure gas.

2. *Carbon Monoxide.*—The gas was prepared by the action of

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[5] *Handbuch der anorganischen Chemie*, vol. ii., part 2, p. 233 (1909).
[6] *Trans.*, xxxix., 630 (1909).
[7] Hofman-Mostowitsch, *Trans.*, xxxix., 637, 638 (1909).
[8] Mostowitsch, *Metallurgie*, vol. vi., No. 14, p. 462 (July 22, 1909).

85-per cent. formic acid (specific gravity, 1.2) upon hot concentrated sulphuric acid.

$$HCOOH + H_2SO_4 = CO + H_2O + H_2SO_4.$$

The sulphuric acid was brought in a gas-bottle to a temperature of from 100° to 150° C., and formic acid was dropped in through a safety thistle-tube. The gas was collected in a gas-holder having water as a seal. It contained 0.58 volume-per cent. of oxygen which had been taken up from the seal-water. The oxygen was determined by passing the gas, dried by means of two wash-bottles charged with caustic potash and concentrated sulphuric acid, through the quartz tube held at from 850° to 900° in the electric-resistance furnace, Fig. 1, and causing the O present to combine with the CO. From the quartz tube the CO with admixed CO_2 was passed through concentrated sulphuric acid and then through two U-tubes charged with soda-lime, which absorbed the CO_2. Thus 3,100 cc. of gas from the holder furnished 0.0641 g. of CO_2; this corresponds to 0.0233 g. of O, which is equal to 17.91 cc. of O at ordinary temperature (20° C.), and corresponds to 0.58 volume-per cent. when referred to the original 3,100 cc.

It was observed that the dry CO with its 0.58 volume-per cent. of O ignited at a temperature lying between 450° and 500° C. This is a lower figure than from 650° to 730°, given by W. Meyer and F. Freyer[9] for a mixture of CO and O, as well as 650° C., given as an average by H. Dixon and H. Coward[10] for moist CO.

3. *Carbon.*—The carbon used as a reducing-agent was chemically-pure sugar-charcoal of Kahlbaum. It was first heated over a Bunsen burner in a covered porcelain crucible, and then in the quartz tube of the electric-resistance furnace, Fig. 1, in a current of nitrogen.

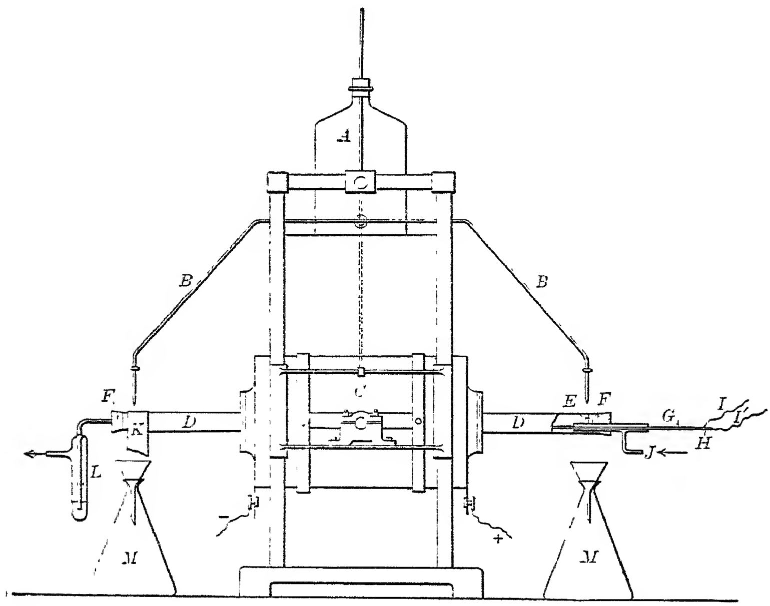
III. Heating-Apparatus.

The furnace used for heating, Fig. 1, was the Heraeus tilting electric-resistance tube-furnace, type E., No. 3, wound to take 24 amperes from a 110-volt circuit. The heating-tube, not

[9] *Zeitschrift für physikalische Chemie*, vol. xi., p. 28 (1893).

[10] *Journal of the Chemical Society, Transactions*, vol. xcv., Part I., p. 519 (1909).

shown, had an inner diameter of 65 mm. and was wound with platinum foil for its entire length of 30 cm.; the large quartz tube, *D* (make of Julius Hulssen & Co., Newcastle-upon-Tyne), was 65 cm. long, had an inner diameter of 20 mm. and a thickness of wall of from 1.7 to 2.0 mm.; the glazed-porcelain boat (not shown) for holding the charge was 7 cm. long, 12 mm. wide, and 7 mm. deep. Temperatures were measured with a Le Chatelier thermo-electric pyrometer and a Siemens-Halske millivoltmeter. The thermo-electric wires, *I* and *I′*, isolated from one another by a Marquardt tube, *H*, were inclosed in a



A. Mariotte flask with cooling-water.
B. Glass tubes.
C. Heraeus tilting electric-resistance tube-furnace.
D. Quartz tube for porcelain boat.
E. Asbestos ring.
F. Rubber stopper.
G. Quartz tube for protection of thermal wires.
I, *I′*. Connection with millivoltmeter.
J. Gas-inlet.
K. Blotting-paper.
L. Railroad-tube.
M. Flasks with funnel for cooling-water dripping from blotting-paper.

Fig. 1.—Heraeus Tilting Electric-Resistance Tube-Furnace.

small quartz tube, *G*, 30 cm. long, 5 mm. inner and 7 mm. outer diameter, which had been drawn out at one end in an illuminating-gas oxygen blow-pipe before closing, in order to obtain a thin wall at the hot-junction. With temperatures exceeding 700° C. the ends of the large quartz tube were cooled by allowing water from the Mariotte flask, *A*, to drip on to blotting paper, *K*, placed over them.

IV. Reduction of Calcium Sulphate by Carbon Monoxide.

In making these tests it was essential to determine the temperature at which reduction began, to ascertain the completion of the reduction, and to prove that no sulphur was lost. The weights of the samples of anhydrous calcium sulphate ranged from 0.1 to 0.4 g. The porcelain boat with its charge was placed in the center of the quartz tube, the air expelled by carbon monoxide, the current turned on, the temperature raised

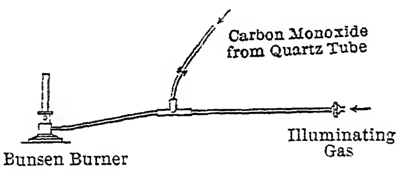


Fig. 2.—Arrangement for Burning Carbon Monoxide from Quartz Tube.

to the desired point, held there for accurately 1 hr., and the current shut off. When the tube had cooled to from 500° to 550° C., in from 10 to 15 min., the boat with its charge was taken out, placed at once in a dessicator containing concentrated sulphuric acid, allowed to cool to room-temperature, and then weighed as quickly as possible because calcium sulphide is readily attacked by the combined action of the moisture and carbonic acid of the air.

The gas issuing from the furnace was conducted through a rubber hose and a T-connection into the tube of a Bunsen burner, as shown in Fig. 2, and burnt with a regulated supply of illuminating-gas. A test required from 3.5 to 5 l. of carbon monoxide.

The results of all experiments were determined quantitatively by loss of weight, but are recorded in Table I. in terms of oxygen. The progress of the experiments carried on at

temperatures ranging from 600° to 750° was ascertained qualitatively by the use of phenolphthalein- and lead-papers. The products of the tests carried on at high temperatures gave off readily hydrogen sulphide when exposed to the air, and did

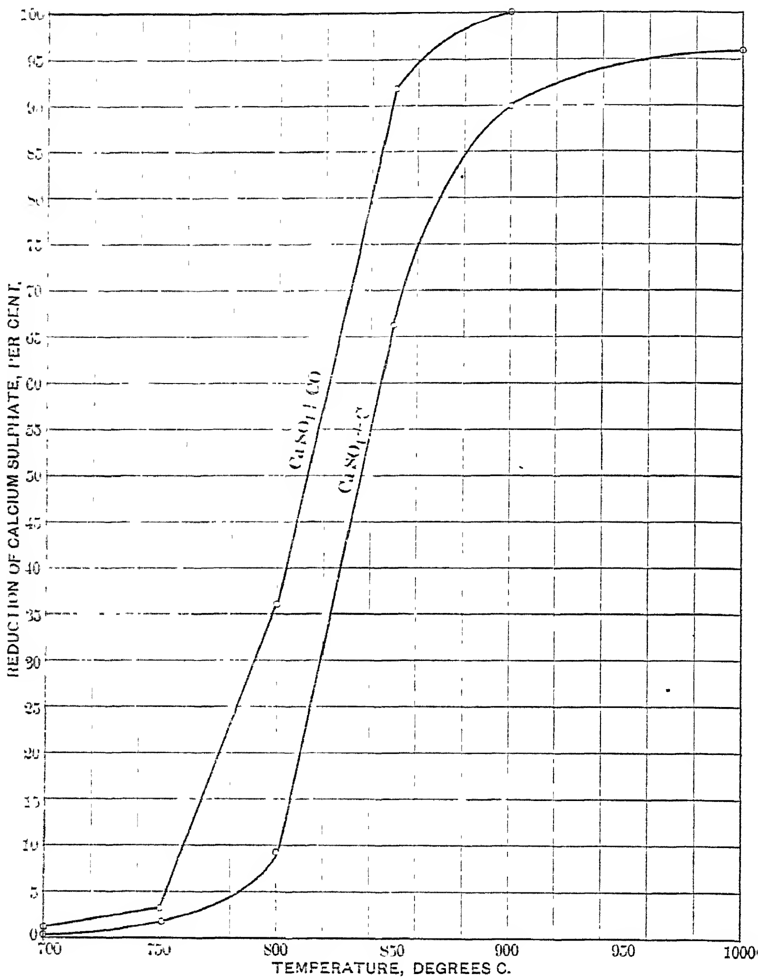


Fig. 3.—Reduction of Calcium Sulphate by Carbon Monoxide and Carbon.

not require such delicate tests; the well-known reaction on sheet-silver was sufficient to prove the presence of calcium sulphide. The reaction that takes place in the reduction of a charge is expressed by $CaSO_4 + 4\ CO = CaS + 4\ CO_2$. Loss

of weight expressed in terms of oxygen was chosen as the criterion instead of the determination of the carbon dioxide set free, as the carbon monoxide contained small amounts of oxygen which would be converted into carbon dioxide. In the qualitative tests, phenolphthalein served to show by the rose coloring the basic reaction of the reduced substance when moistened with water: $CaS + 2\,H_2O = Ca(OH)_2 + H_2S$, and the lead-paper by a yellowish to brownish coloring the sulphurizing action of the calcium sulphide formed, $PbCO_3 + CaS = PbS + CaCO_3$. Whenever necessary, the progress of these reactions was followed by examination under a miscroscope of 56 diameters: two papers were spread on the stage, a grain of reduced substance placed on each, and then moistened with water from a capillary tube; a pale rose-color, and a yellow-brownish ring proved the presence of any calcium sulphide.

The absence of sulphur dioxide in the gases issuing from the quartz tube was ascertained by the use of collection-tubes, Fig. 5, charged with potassium permanganate and barium chloride solution (see p. 776). Thus 0.2320 g. of $CaSO_4$ gave, upon heating in CO to from 950° to 1,000° for from 1 to 1.5 hr., 0.1232 instead of the theoretical 0.1230 g. of CaS, and the permanganate was not decolorized nor was the barium chloride rendered turbid. Calcium sulphate is therefore reduced by carbon monoxide to calcium sulphide without any loss of sulphur. The calcium sulphide is a yellowish sintered product. It is not changed at ordinary temperature (20°) in dry air, and is not attacked by dry carbon dioxide at temperatures reaching 400°; higher temperatures were not tried in the present work.

The results obtained are assembled in Table I. and represented graphically in Fig. 3.

TABLE I.—*Reduction of Calcium Sulphate by Carbon Monoxide.*

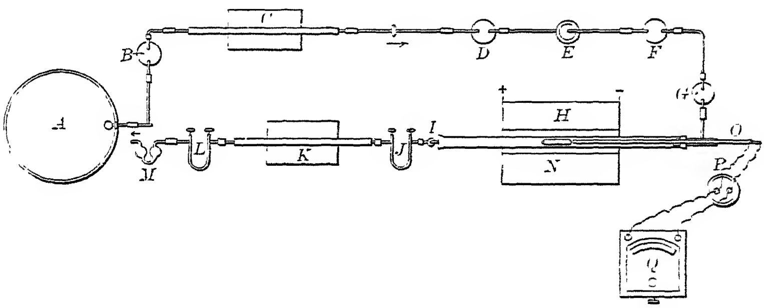
Sample No.	Weight of Substance, $CaSO_4$, before Reduction	Temperature of Reduction	Time of Reduction	Weight of Substance after Reduction	Weight of Oxygen in Substance, $CaSO_4$, before Reduction	Loss of Oxygen by Reduction	Reduction of Calcium Sulphate	
	Grams	Degrees C.	Hours	Grams	Grams	Grams	Per Cent	
1	0.2000	500	1	0.2000	0.0940	....	....	
2	0.2000	600	1	0.2000	0.0940	....	....	
3	0.2000	650	1	0.2000	0.0940	....	....	
4	0.2000	670–680	1	0.2000	0.0940	....	....	
5	0.2000	700	2	0.1989	0.0940	0.0011	1.17	1.05
6	0.4602	700	1	0.4582	0.21622	0.0020	0.925	
7	0.1152	750	1	0.1134	0.05413	0.0018	3.33	3.27
8	0.1993	750	1	0.1963	0.09366	0.003	3.20	
9	0.1993	800	1	0.1711	0.09366	0.0282	30.11	36.29
10	0.1306	800	1	0.1046	0.06137	0.0260	42.47	
11	0.2000	850	1	0.1100	0.0940	0.0900	95.76	91.79
12	0.2067	850	1	0.1267	0.09713	0.0800	82.41	
13	0.1620	850	1	0.0880	0.07613	0.074	97.20	
14	0.1374	900	1	0.0728	0.06457	0.0646	100	100
15	0.1374	900	2	0.0728	0.06457	0.0646	100	
16	0.2000	900	1	0.1059	0.0940	0.0941	100	
17	0.6336	900	1	0.3357	0.2978	0.2978	100	

Table I. shows that up to 680° C. calcium sulphate remains unattacked by carbon monoxide, that the reduction at 700° reaches 1.05 per cent., that it increases very rapidly with rise of temperature, is complete at 900°, and nearly so at 850°. The reduction therefore begins between 680° and 700°. Fig. 3 brings out clearly the quickness of the reaction between 750° and 850°. The discrepancies between the duplicate tests in the table are due to two causes. The current was drawn from a main serving the building, containing many users; while fluctuations were immediately corrected by altering the resistance, variations in temperature of from 10° to 15° C. for from 3 to 5 min. could not be avoided. The other cause is due to the difference in time required to bring a cold or a warm furnace to the required temperature. Starting with a cold furnace, it took from 1 to 1.5 hr. to bring the quartz tube to 800°; with a warm furnace recording from 200° to 300° it took only from 30 to 40 min. to attain the same end. Hence, in the first case the charge was exposed for a longer time to a temperature above 680° than in the second.

V. Reduction of Calcium Sulphate by Carbon.

The reduction of calcium sulphate by solid carbon was carried on in an atmosphere of nitrogen, and the necessary heat was furnished by the Heraeus furnace.

Nitrogen was prepared according to the method given by Knorre[11] from potassium nitrite and ammonium sulphate. The gas-bottle was charged with 1 part by weight of a concentrated solution of potassium nitrite and about 2.5 parts of a similar solution of ammonium sulphate; the mixture was made ammoniacal and heated. The nitrogen generated was freed from ammonia by passing through dilute sulphuric acid and collected in a gas-holder having a water-seal.



A. Gas-holder for nitrogen.
B. Wash-bottle with sulphuric acid.
C. Combustion-furnace with glass tube charged with metallic copper.
D. Wash-bottle with caustic potash.
E. Calcium chloride tower.
F and *G.* Wash-bottles with sulphuric acid.
H. Heraeus electric-resistance tube-furnace.
I. Railroad-tube with sulphuric acid.
J. U-tube with soda-lime.
K. Combustion-furnace with glass tube charged with cupric oxide.
L. U-tube with soda-lime.
M. Glass bulb with sulphuric acid.
N. Porcelain boat.
O. Thermo-electric couple in quartz tube.
P. Cold-junction.
Q. Siemens-Halske millivoltmeter.

Note: parts *J*, *L*, and *M* are drawn in elevation.

Fig. 4.—Arrangement of Apparatus for Reduction of Calcium Sulphate by Solid Carbon.

[11] *Die chemische Industrie*, vol. xxv., pp. 531, 550 (1902).

The general arrangement of apparatus used in the experiments is shown in Fig. 4. It is similar to that employed in the carbon-monoxide tests, excepting that the nitrogen, dried by concentrated sulphuric acid in *B*, was freed from accompanying oxygen by passing over metallic copper kept at a red heat in combustion-furnace, *C*, and then freed from carbon dioxide and water by passing through concentrated caustic potash, *D*, calcium chloride, *E*, concentrated sulphuric acid, *F* and *G*, before entering the electric furnace, *H*. Before every test the copper—granules and wire screenings—was brought to a red heat in a current of hydrogen and thus any oxide present was reduced to the metallic state.

The charges for the porcelain boat, *N*, were made up to correspond to the formula $CaSO_4 + 4\ C$; a fresh charge was prepared for every test, and the constituents were rubbed together in an agate mortar.

The reduction of calcium sulphate may take place according to the reactions

$$CaSO_4 + 4\ C = CaS + 4\ CO$$
$$CaSO_4 + 2\ C = CaS + 2\ CO_2.$$

Hence provision had to be made to determine both CO_2 and CO. The gases leaving the quartz tube were dried in railroad-tube, *I*, charged with concentrated sulphuric acid, passed through U-tube, *J*, charged with soda-lime to take up CO_2, and then through a glass tube charged with granulated cupric oxide and held at about 600° C. in combustion-furnace, *K*, to oxidize the CO to CO_2, which was then absorbed by a second U-tube, *L*, charged with soda-lime; finally the nitrogen passed through a glass bulb, *M*, charged with concentrated sulphuric acid, into the open.

In making a test, the porcelain boat with its charge was placed in the quartz tube, the furnace being cold, and then the air replaced by nitrogen; next the soda-lime tubes were put in position, nitrogen was forced through the whole train for 15 min., the U-tubes were taken out, weighed, and put again in place. Everything being ready, the current was turned on, the furnace brought to the desired temperature and held there for exactly 1 hr. The current was shut off, the rubber con-

nections of the U-tubes were clamped, the tubes removed to the scale-case, and weighed 10 min. later.

As it is very difficult to remove from nitrogen all the oxygen, and as an experiment required from 8 to 10 l. of gas, the simplest way of getting around the difficulty was to apply a correction. The necessary data were found by making with each reduction-test a blank-test with sugar-charcoal, determining the CO_2 and CO formed and deducting the figures from those obtained in the reduction-experiments. From the corrected values of CO_2 and CO the total O was calculated, the corresponding amount of $CaSO_4$ calculated, and this referred back in per cent. to the $CaSO_4$ weighed out at first.

The progress of the reduction of calcium sulphate was ascertained by phenolphthalein- and lead-papers, as in the carbon-monoxide experiments. With the latter the yellowish coloring of the paper, assisted by the addition of a drop of dilute sulphuric acid, $CaS + H_2SO_4 = CaSO_4 + H_2S$, became visible only after the carbonaceous residue had been removed.

The results are given in Table II., and represented graphically in Fig. 3.

TABLE II.—*Reduction of Calcium Sulphate by Carbon.*

Sample No.	Anhydrous Calcium Sulphate.	Sugar-Charcoal.	Oxygen in Calcium Sulphate	Temperature of Reduction.	Time of Reduction.	CO_2 Obtained in Reduction.	CO Obtained in Reduction.	Oxygen Contained in CO_2 and CO.	CO_2 Obtained.	CO Obtained.	Calcium Sulphate Reduced.	
	Grams.	Grams.	Grams.	Deg. C.	Hr	Grams	Grams.	Grams.	Per Ct.	Per Ct.	Grams.	Per Ct
18	0.2973	0.1057	0.1397	600	1	.....	.....	.....	.....	.....	.....	.....
19	0.2973	0.1057	0.1397	650	1	.....	.....	.....	.....	.....	.....	.....
20	0.2973	0.1057	0.1397	700	1	.....	0.001	0.00057	.....	100.0	0.00121	0.41
21	0.2980	0.1057	0.1402	750	1	0.003	0.0004	0.0024	88.2	11.8	0.00512	1.71
22	0.3000	0.1057	0.14098	800	1	0.018	.....	0.0131	100.0	.....	0.02788	9.29
23	0.3000	0.1057	0.14098	850	1	0.1277	0.0018	0.0939	98.6	1.4	0.1998	66.6
24	0.3000	0.1057	0.14098	900	1	0.1672	0.0088	0.1266	95.0	5.0	0.2694	89.9
25	0.3000	0.1057	0.14098	1,000	1	0.1675	0.0212	0.1337	88.76	11.24	0.2845	94.83
26	0.3027	0.1057	0.14225	1,000	1	0.1687	0.02564	0.1374	86.83	13.17	0.2924	96.59

Table II. shows that the reduction of calcium sulphate begins at 700° C. and is practically complete at 1,000°; the curve visualizes the fact that between 800° and 900° C. the reduction progresses a great deal more quickly than at temperatures that lie below or above this range. From the relative proportions of CO_2 and CO it is seen that at 800° the reduction takes place according to the formula $CaSO_4 + 2\ C = CaS + 2\ CO_2$, and

that above this temperature the reaction $CaSO_4 + 4 C = CaS + 4 CO$ begins to assert itself. The same was observed by Mr. Mostowitsch[12] in the reduction of barium sulphate by carbon.

A comparison of the reduction of calcium sulphate by carbon monoxide and carbon shows that the former acts more readily and hence more effectively than the latter. This was to be expected, as the gaseous carbon monoxide penetrates the porous charge and attacks it at all points, while it is impossible to obtain a similar intimate contact between two solids, especially where their specific gravities and volumes are as far apart as those of calcium sulphate and amorphous carbon. While the mixture of the two ingredients, made intimate by rubbing in an agate motor, appeared homogeneous to the naked eye, when examined under a microscope of a magnification of 125 diameters, the carbon was found to be unevenly distributed through the calcium sulphate, forming here and there nest-like aggregations. This circumstance may explain the irregularity of composition of the gas obtained below 800°, as $CaSO_4$ in contact with an excess of C would form mainly CO, while CO_2 would preponderate under reverse conditions. At temperatures above 800° there is more regularity in the ratio $CO_2 : CO$ than below, as the tendency of C to burn to CO increases with the rise of temperature. A microscopical examination of the samples reduced above 800° showed them to be grayish, fritted, porous, coke-like masses, which appeared the lighter in color the higher had been the temperature of reduction. Well-reduced samples gave off hydrogen sulphide accompanied by effervescence when treated with hydrochloric or sulphuric acid.

VI. Oxidizing-Roast of Calcium Sulphide.

Having reduced calcium sulphide from calcium sulphate by means of carbon monoxide and carbon, it seemed of general as well as metallurgical interest to reverse the process and study the behavior of calcium sulphide when subjected under controlled conditions to an oxidizing-roast in pure dry air.

The apparatus used was the same as in the preceding experiments; the manner of supplying pure dry air under pressure has been given in our former paper.[13]

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[12] *Metallurgie*, vol. vi., No. 14, p. 462 (July 22, 1909).

[13] *Trans.*, xxxix., 635 (1909).

In the first test 0.2 g. of $CaSO_4$ was reduced by a current of CO in 1 hr. at a temperature of 900° C. and gave 0.1059 g. of CaS; the experiment was repeated for an additional hour, which left the weight of 0.1059 g. unchanged. This corresponds to a loss in O of 0.0941 g., while theory calls for 0.0940 g.; the CaS was therefore chemically pure. The 0.1059 g. of CaS was next subjected to an oxidizing-roast, first at 800°, then three consecutive times at 900°, until the constant weight of 0.1823 g. was obtained. Table IV. gives the numerical data in detail.

TABLE IV.—*Oxidizing-Roast of Calcium Sulphide.*

Sample No.	Weight of Substance Before Oxidation.	Temperature of Oxidation	Time of Oxidation.	Weight of Substance After Oxidation.	Increase in Weight, Equals Oxygen Taken Up.
	Grams	Degrees C.	Hours.	Grams.	Grams.
27	0.1059 (CaS)	800	1	0.1419	0.0360
28	0.1419	900	1	0.1809	0.0390
29	0.1809	900	1	0.1823	0.0014
30	0.1823	930–950	1	0.1823	.. ..
				Total..............	0.0764

The roasted charge gave a strong alkaline reaction with phenolphthalein-paper; under the microscope only a slight tarnishing of the lead-paper was observed, showing that only a minute quantity of the original calcium sulphide had remained undecomposed. Neglecting this trace of calcium sulphide and assuming that the product consists of calcium sulphate and calcium oxide, their relative proportions, as well as the loss in sulphur, were easily calculated.

The difference in weight between the product and the original substance represents the amount of O taken up, or 0.1823 — 0.1059 = 0.0764 g. of O, which has to be apportioned between $CaSO_4$ and CaO. Let $X =$ g. of $CaSO_4$ and $Y =$ g. of CaO, then O-content in X g. $CaSO_4$ is $\frac{64}{136.19}X$, and that in Y g. CaO is $\frac{16}{56.13}Y$. Now, $X + Y = 0.1823$ g., and $\frac{64}{136.19}X + \frac{16}{56.13}Y = 0.0764$ g. Solving the equation gives

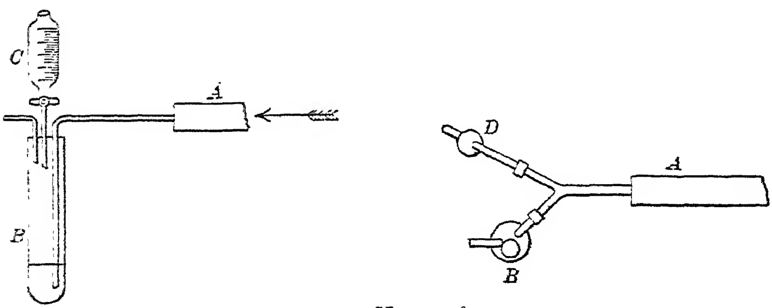
$X = 0.13215$ g., or 72.49 per cent. of $CaSO_4$

$Y = 0.0502$ g., or 27.51 per cent. of CaO

$X + Y = 0.18235$ g., or 100.00 per cent. of oxidized product.

If the oxidation of calcium sulphide had taken place according to the reaction $CaS + 4O = CaSO_4$, the 0.1059 g. of CaS ought to have furnished 0.2 g. of $CaSO_4$; but the product weighed only 0.1823 g., hence there was a loss; and this was sulphur, as proved by the subsequent experiments. Its amount is easily calculated. The 0.1059 g. of CaS contains 0.04703 g. of S; the 0.13215 g. of $CaSO_4$ contains 0.03111 g. of S; the difference, 0.01592 g., represents the S lost, or referred to the sulphur-content of the original 0.1059 g. of CaS gives a loss of 33.85 per cent. of sulphur.

The following test was made, to prove that the loss in weight is due to the liberation of sulphur, to ascertain again the proportion of calcium sulphate and calcium oxide in the product, and to determine the amount of undecomposed sulphide.



A. Quartz tube of Heraeus furnace.
B. Test-tube with potassium permanganate.
C. Graduated pipette.
D. Railroad-tube with barium chloride.

FIG. 5.—APPARATUS FOR TESTING FOR SULPHUR DIOXIDE.

The apparatus used for proving the liberation of sulphur is shown in Fig. 5. The gases from the quartz tube, *A*, closed by a perforated rubber stopper, were conducted alternately through the two absorption-vessels, *B* and *D*; vessel *D* is a railroad-tube containing barium chloride acidulated with hydrochloric acid and charged with bromine to oxidize sulphurous to sulphuric acid, which the barium chloride will precipitate; vessel *B* is a wide test-tube filled with 15 cc. of dilute sulphuric acid (1 : 10) and holding in the perforated rubber stopper a graduated separatory funnel containing a solution of potassium permanganate (1 : 100), which drips into the vessel, *B*, and is decolorized by sulphurous acid.

In the test the porcelain boat, charged with 0.2440 g. CaS, was introduced into the quartz tube, the furnace heated at first to from 300° to 350°, the temperature raised in 10-min. intervals for 50°, and any change in the absorption-tubes noted. The current of air was made to pass the apparatus at a quick speed in order to insure the expulsion of the sulphurous acid in the interval of 10 min. The observations made are recorded in Table V.

TABLE V.—*Tests for Sulphurous Acid.*

Temperature.	Permanganate Solution.	Barium Chloride Solution.
Degrees C.		
350–450	No discoloration.	No turbidity.
450–550	Faint discoloration.	No turbidity.
550–650	Faint discoloration.	No turbidity.
650–750	Slight discoloration.	No turbidity
800	Quick discoloration.	Light turbidity.
850	Instantaneous discoloration.	Heavy precipitate.
900	Instantaneous discoloration.	Heavy precipitate.

Table V. shows that decomposition became noticeable at about 500° and was active at 800°. Experiments made in another connection had shown that the reaction with barium chloride was very delicate, 0.6 mg. of SO_2 causing a solution to become decidedly turbid. Therefore 800° may be accepted as denoting the practical beginning of evolution of sulphur dioxide.

The heating was carried on until the sample, held for from 1.5 to 2 hr. at from 900° to 950°, ceased to cause any change in the absorption-liquids, which were removed every 10 or 15 min., thus proving that all the volatile sulphur had been expelled. The roasted product weighed 0.4225 g.; deducting the original weight of 0.2440 g. gives an increase of 0.1785 g. If the oxidation had taken place according to the equation $CaS + 4\,O = CaSO_4$, the weight of the product ought to have been 0.2160 g. A qualitative analysis of a grain of the product gave with phenolphthalein-paper a decided rose coloring, and with lead-paper a visible yellow stain, showing that there was present much calcium oxide and little calcium sulphide.

For the determination of the residual sulphide sulphur, a modification of the method given by L. Campredon[14] for deter-

[14] *Guide Pratique du Chimiste Métallurgiste et de l'Essayeur*, p. 551 (Tignol, Paris, 1898).

mining sulphur in pig-iron was used. The remaining 0.4035 g. of the roasted product was treated with boiling water, hydrochloric acid was added, and simultaneously a standardized solution of iodine: the excess iodine was titrated with sodium hyposulphite, using starch solution as an indicator.

The 0.40335 g. gave 0.0008 g. of sulphide sulphur = 0.001802 g. or 0.446 per cent. of CaS: this gives for the original 0.4225 g. of roasted product, 0.00189 g. of CaS.

With the data thus obtained, the composition of the roasted product is readily found. For the experiment there was used 0.2440 g. of CaS: there remained undecomposed 0.00189 g. of CaS, hence there was oxidized 0.24211 g. of CaS, and this furnished 0.4225 — 0.00189 = 0.42061 g. of roasted product consisting of $CaSO_4$ and CaO: the difference in weight between the product and the original substance represents the amount of O taken up, or 0.42061 — 0.24211 = 0.1785 g. of O, which has to be apportioned between $CaSO_4$ and CaO. Proceeding with equations of two unknowns as before, we obtain as composition of the roasted product:

$CaSO_4$	0.3170 g. or	75.03 per cent.
CaO	0.10361 g. or	24.52 per cent.
CaS	0.00189 g. or	0.446 per cent.
Total,	0.42250 g. or	100.000 per cent.

Calculation of the loss of sulphur gives the following: The original 0.2440 g. of CaS contained 0.10836 g. of S; in the roasted product are present 0.3170 g. of $CaSO_4$ with 0.07461 g. of S, and 0.00189 g. of CaS with 0.0008 g. of S, or a total of 0.07541 g. of S, which represents a loss of 0.10836 — 0.07541 = 0.03295 g. of S, or, referred to the original 0.10836 g. of S, a loss of 30.41 per cent.

The results of both tests are given in Table VI.

TABLE VI.—*Oxidizing-Roasts of Calcium Sulphide.*

Sample No.	Weight of Substance Before Roasting.		Composition of Substance After Roasting.			Sulphur-Content of Roasted Substance.	Loss of Sulphur in Roasting.
	CaS.	S.	CaS.	CaO.	$CaSO_4$.		
	Grams.	Grams.	Per Cent.	Per Cent.	Per Cent.	Grams.	Per Cent.
31	0.1059	0 04703	trace.	27.51	72.49	0 01592	33.85
32	0.2440	0.10836	0.45	24.52	75.03	0.03295	30.41
		Average.		26.3	73.7		32.13

As in the roasting of calcium sulphide there ought not to be any loss in sulphur, if the reaction proceeds according to equation $CaS + 4\ O = CaSO_4$, the great loss of 32.13 per cent. shown by the experiments must be due to some secondary reaction between calcium sulphate and calcium sulphide. This led to the following experiments:

VII. Interaction of Calcium Sulphate and Calcium Sulphide at Elevated Temperatures.

Reactions between metallic sulphates and sulphides at elevated temperatures form the bases of several metallurgical processes. Thus in smelting galena-ore in the reverberatory furnace or ore-hearth the reactions $PbSO_4 + PbS = 2\ Pb + 2\ SO_2$ and $3\ PbSO_4 + PbS = 4\ PbO + 4\ SO_2$ are of importance. Recent studies of this subject are by Lodin[15] and R. Schenk;[16] Sakur[17] has studied the reaction $Ag_2S + Ag_2SO_4 = Ag_4 + 2\ SO_2$; Ingalls[18] notes the interaction of $BaSO_4$ and BaS at the temperature of the electric arc; Lepiarczyk[19] assumes an action of $CaSO_4$ upon CaS to explain the loss of 32.86 per cent. of S he experienced in reducing $CaSO_4$ by C. That this reaction cannot take place under reducing conditions will be shown later; however, his figure for loss of S agrees well with ours of 32.13 per cent.

In the experiments made to obtain data to prove the sulphate-sulphide reaction $3\ CaSO_4 + CaS = 4\ CaO + 4\ SO_2$, the charges were made up to correspond to the above formula; the components were rubbed together in an agate mortar, and then the mixture, transferred to a porcelain boat, was heated in the electric furnace in a current of nitrogen. The gas issuing from the quartz tube was tested for sulphur dioxide by potassium permanganate and barium chloride solutions, as in preceding experiments. At 750° C. the permanganate solution was slightly decolorized, while barium chloride remained clear; at 800° the former reagent was quickly decolorized and the latter showed a decided turbidity. Therefore at 800° the

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[15] *Comptes rendus*, vol. cxx., No. 21, pp. 1164 to 1167 (1895).
[16] *Physikalische Chemie der Metalle*, Knapp, Halle a/S., p. 181 (1909).
[17] *Berichte der deutschen chemischen Gesellschaft*, vol. xli., p. 3356 (1908).
[18] *Metallurgy of Zinc and Cadmium*, p. 685, footnote (1903).
[19] *Metallurgie*, vol. vi., No. 13, pp. 414, 415 (July 8, 1909).

reaction of $CaSO_4$ and CaS begins, and between 850° and 900° it becomes energetic and is accompanied by an abundant evolution of SO_2. The results obtained with two charges are given in Table VII.

TABLE VII.—*Interaction of Calcium Sulphate and Calcium Sulphide.*

Sample No	Composition of Mixture.		Sulphur-Content of Mixture.	Loss in Weight by Heating $= SO_2$	Loss in S by Heating.	
	$CaSO_4$.	CaS.				
	Grams	Grams.	Grams.	Grams.	Grams.	Per Cent
33	0.9497	0.1678	0.29726	0.0973	0.0487	16.39
34	0 8150	0.1440	0.2550	0.1496	0.0748	29 33

It was not possible to set free all of the sulphur. Sample No. 33 was held for from 30 to 40 min. at 900° C., sample No. 34 for 90 min. at from 950° to 1,000°, and the respective eliminations were only 16.39 and 29.33 per cent., instead of 100 per cent. as called for by the formula. The reason for this imperfection lies in the impracticability of bringing the two acting solid substances into absolute intimate contact. Just as was the case with the reduction of $CaSO_4$ by C, local accumulations of one or the other reagent could be recognized. Further, when contiguous particles of $CaSO_4$ and CaS have acted upon one another there is formed between them a barrier of CaO which prevents further contact and stops the reaction. Removing the charge from the boat and rubbing it in an agate mortar to furnish new points of contact was not feasible on account of the changes CaS and CaO would undergo if exposed to air containing H_2O and CO_2.

The imperfect elimination of sulphur in roasting calcium sulphide may be explained in a similar manner: the $CaSO_4$ formed acts upon undecomposed CaS, forming CaO, which again prevents the necessary contact for further chemical action. The greater liberation of SO_2 by roasting CaS as compared with the interaction of mixed $CaSO_4$ and CaS is of course due to air penetrating the porous charge more perfectly than is possible with the mixed solids, and thus forming $CaSO_4$ throughout the mass, which being in approximately molecular contact with undecomposed CaS causes the reaction to proceed more vigorously.

A microscopic examination of single grains frequently revealed a kernel of CaS inclosed by a rind of $CaSO_4$ and CaO, a phenomenon characteristic in kernel-roasting of suitable sulphide copper-ore.

VIII. Summary of Results.

The results obtained in the investigation are the following:

1. Calcium sulphate is reduced both by carbon monoxide and by carbon to calcium sulphide without loss of sulphur. The reduction by carbon monoxide begins at 700° C. and is finished at 900°. The reduction by carbon in a neutral atmosphere begins at 700° and can be considered finished at 1,000°; at the lower end of the temperature-range the reducing carbon burns mainly to carbon dioxide, at the higher in part to carbon monoxide. The reduction by carbon is more difficult to accomplish than by carbon monoxide on account of the impossibility of obtaining an equally intimate contact of the reagents.

2. Calcium sulphide is unchanged at ordinary temperature in pure dry air and carbon dioxide; carbon dioxide has no effect up to 400°.

3. Roasting calcium sulphide in pure dry air forms a product consisting of 76 per cent. of $CaSO_4$ and 26 per cent. of CaO; the process is accompanied by a loss of 32 per cent. of S, caused by the interaction of $CaSO_4$ and CaS. This interaction takes place in a neutral as well as in an oxidizing, but not in a reducing, atmosphere; it begins at about 800° and is intensified by a rising temperature, but is not complete, on account of the CaO produced forming a barrier between the active agents.

IX. Behavior of Calcium Sulphide in Some Metallurgical Processes.

In a discussion of the metallurgical behavior of calcium sulphide the start has to be made with calcium sulphate, which is formed in the roasting of calcareous sulphide ores, or occurs as gangue in the ore, or is added to the charge as a sulphurizing flux.

The recent experiments of Lepiarczyk[20] have shown that in roasting calcareous blende the lime present is converted almost

[20] *Metallurgie*, vol. vi., No. 13, p. 414 (July 8, 1909).

quantitatively into calcium sulphate. The generally-accepted reduction of calcium sulphate to sulphide in the subsequent distillation in a retort is questioned by him; he states that the reaction $CaSO_4 + 4\ C = CaS + 4\ CO$ is not quantitative, and assumes that in the reducing atmosphere of the retort part of the calcium sulphate is decomposed by calcium sulphide according to $3\ CaSO_4 + CaS = 4\ CaO + 4\ SO_2$. Our experiments prove that both of these ideas are wrong.

As regards the action of solid carbon, they have shown that 96 per cent. of the $CaSO_4$ is reduced without any loss of sulphur. In order to confirm our results, a new experiment was made and the actual sulphide sulphur compared with the theoretical. In it 0.4363 g. of mixture of $CaSO_4 + 4\ C$ was heated in nitrogen for 1.5 hr. at a temperature rising from 500° to 1,000° C., and the issuing gas tested as before for SO_2 by means of $KMnO_4$ and $BaCl_2$. At 900° a slight clouding of the $BaCl_2$ appeared, but stopped shortly, and did not reappear. The completely reduced sample gave 0.0753 g. of sulphide-sulphur, while theory calls for 0.0759 g., representing a loss of 0.08 per cent. Repeating the test with twice the amount of C, the qualitative test of the gases gave no SO_2 whatever, either with $KMnO_4$ or $BaCl_2$, showing the complete reduction to sulphide in a reducing atmosphere.

As regards the reaction $3\ CaSO_4 + CaS$, our tests have shown that it takes place in a neutral and an oxidizing atmosphere. It cannot take place in a reducing atmosphere, as the $CaSO_4$ is too quickly converted into CaS by CO and C. In large-scale zinc-work reducing carbon to the amount of about 45 per cent. of the weight of the ore is mixed with the latter. The volume of 40 per cent. of reducing carbon is equal to the volume of the ore[21] and therefore a fairly intimate contact of the components of the charge is assured. The atmosphere in a retort is reducing, as the issuing gas contains[22] from 95 to 97 per cent. of CO and not more than from 0.9 to 1.0 per cent. of CO_2. If the sulphate-sulphide reaction did take place in the retort, the gases ought to contain SO_2, which they do not;[23] the blue powder ought to contain some sulphate, which

[21] W. R. Ingalls, *Metallurgy of Zinc and Cadmium*, pp. 203, 507 (1903).
[22] Ingalls, *op. cit.*, p. 204.
[23] Firket, in Ingalls, *op. cit.*, p. 204.

it does not,[24] but only 0.05 per cent. of S.[25] Possible reactions between CaS and ZnO [26] lie outside of the scope of the present paper.

In smelting in the blast-furnace ores containing calcium sulphate in the gangue, calcium sulphide can be formed only in a reducing atmosphere. It may remain undecomposed and to some extent enter the matte, but will be taken up mainly by the slag; it may react with metal or metallic oxide, forming metallic sulphide and calcium oxide, the latter combining with the silica of the slag. Thus in the strongly-reducing atmosphere of the iron blast-furnace the calcium sulphide, formed by the reduction of calcium sulphate or by the interaction of calcium carbonate with metallic sulphide, enters the slag to an extent of 7 per cent.[27]

As the reactions $CaSO_4 + 4\ CO = CaS + 4\ CO_2$ and $CaSO_4 + 2\ C = CaS + 2\ CO_2$ are irreversible, carbon dioxide having no decomposing effect upon calcium sulphide, and as the ferrous slags formed in smelting raw or roasted lead-ores containing calcium sulphate hold considerable amounts of calcium sulphide, it appears that calcium sulphate is reduced even in the mildly-reducing atmosphere of a lead blast-furnace. The presence of calcium sulphide in matte is shown by the precipitation-experiments of Schütz.[28] In fusing PbS with Fe and CaS, the yield in lead was decreased 40 per cent. below the amount obtained in the absence of CaS, and the matte-fall increased by a content of 24 per cent. of CaS.

In a strictly reducing fusion of sulphide copper-ore containing calcium sulphate, the calcium sulphide formed is found to some extent in the matte, but mostly in the slag.[29] The above statements of practical facts prove that it is justifiable to assume that calcium sulphide remains unchanged at elevated temperatures.

In smelting in the blast-furnace ores to which gypsum is added as a sulphurizing-agent, the conditions have to be such

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[24] Ingalls, *Metallurgy of Zinc and Cadmium*, p. 529 (1903).
[25] Private Notes from Stolberg, Rhenish Province, Prussia.
[26] Ingalls, *op. cit.*, pp. 207, 208; Lepiarczyk, *Metallurgie, op. cit.*, p. 417.
[27] A. Ledebur, *Handbuch der Eisenhüttenkunde*, 4th ed., p. 592.
[28] *Metallurgie*, vol. iv., No. 20, p. 697 (Oct. 22, 1907).
[29] A. Baikoff, Fonte des minérais de cuivre pour matte aux fours à cuve, *Annales de l'Institut Polytechnique de St. Petersbourg*, vol. ix., pp. 608, 609 (1908).

as to insure a reduction to sulphide, and to prevent expulsion of sulphur as sulphur dioxide. These are a reducing atmosphere, an intimate contact of ore with carbon, and a correct formation-temperature of slag. In the smelting of oxide nickel-ore with gypsum as sulphurizing-agent and with little iron, it appears necessary to have a high blast-furnace, a large amount of coke (from 25 to 30 per cent. of weight of charge), a slag low in iron that forms at an elevated temperature (SiO_2, 50 per cent.; and from 1,400 to 1,500° C. at tuyère-section), an intimate contact of charge and reducing carbon (requiring briquetting), and a blast of moderate pressure. The gases from such a furnace taken 6 ft. below the throat gave Mostowitsch, CO_2, 11.0; CO, 16.4; O_2, 0.5; and N, 72.1 per cent. by volume. It is essential that the calcium sulphate be reduced by carbon monoxide and carbon in the upper part of the furnace at a temperature between 800° and 1,000° C., and the calcium sulphide react at once with the metallic oxide: $CaS + MO = CaO + MS$. If this is not the case, and the CaS enters the fusion-zone below, it will be taken up by the slag as such or as CaO after it has been oxidized by the blast, any $CaSO_4$ formed having been decomposed by SiO_2, and the S liberated as SO_2. The conditions given for reduction and sulphurization require that CaS and MO act upon one another in the solid state. This has not been proved as yet, but may be assumed to be true, considering that it has been shown first by Day and Shepherd[30] and confirmed later by others, that the formation-temperature of a silicate, contrary to general belief, lies lower than the melting-temperature of the formed silicate. Calcium sulphide is a powerful sulphurizing-agent which acts at low temperatures. The reactions between it and metallic oxide or sulphate are strongly exothormic:

$$PbSO_4 + CaS = PbS + CaSO_4 + 27{,}600 \text{ cal.}$$
$$PbO + CaS = PbS + CaO + 6{,}600 \text{ cal.}$$
$$FeSO_4 + CaS = FeS + CaSO_4 + 12{,}200 \text{ cal.}$$
$$CuSO_4 + CaS = CuS + CaSO_4 + 51{,}500 \text{ cal.}$$

Doeltz[31] has shown that the first of these reactions takes

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[30] *Journal of the American Chemical Society*, vol. xxviii., No. 9, pp. 1,089 to 1,114 (Sept.,1906).

[31] *Metallurgie*, vol. ii., No. 19, p. 460 (Oct. 8, 1905).

place at a low temperature. Qualitative tests with PbO, $FeSO_4$, $CuSO_4$ and Cu_2Cl_2 proved the chemical activity of CaS at a low temperature. These compounds, mixed severally with CaS and warmed gently over a Bunsen burner, reacted quickly, often accompanied by incandescence. The original whitish or pale-yellow mixtures became brown to black, and examination of the products showed that they contained metallic sulphide and calcium sulphate, respectively oxide or chloride. The original mixtures all gave off some hydrogen sulphide, due to the action of the moisture of the air; using a slight excess of metallic compound over that required by the formula, all evolution of hydrogen sulphide by the product in moist air ceased.

From the aforesaid it is clear that, in smelting in the blast-furnace an oxide ore with gypsum as flux, the matte to be produced must be formed, mainly in the upper part of the furnace, by reactions taking place while the charge is still solid; lower down, when the matte begins to fuse, it will trickle through the slag that is still forming and then join the melted slag. During its downward passage it will change in composition, becoming richer in the available metal that has a stronger affinity for sulphur.

Gypsum is desirable as a sulphurizing flux only with fusions in which there is an objection to the use of pyrite, which with its sulphur adds iron, and usually some copper, arsenic, antimony, etc., to the charge. Further disadvantages of gypsum are that it requires special reducing blast-furnace conditions, and that it consumes a considerable amount of carbon for its own reduction to calcium sulphide.

Labor-Saving Appliances in the Assay-Laboratory.

BY EDWARD KELLER, PERTH AMBOY, N. J.

(Canal Zone Meeting, November, 1910.)

UNDER the title, Labor-Saving Appliances in the Works-Laboratory, I published a paper[1] in which was described how multi-manipulations in a works-laboratory and in the furnace-room of an assay-laboratory, can be condensed into single manipulations by applying the proper mechanical devices. For example, I now deposit in, or withdraw from, a muffle a set of 48 cupels as one unit. The front row (8) of these are blanks or heaters; the other 40, when they have attained the proper temperature in the muffle, are charged simultaneously with the 40 lead-buttons. Three manipulations here accomplish an operation which by the generally customary method requires 136, and this number is only limited by the size of muffle and cupels. This operation is an enlargement of what I have already described in my former paper, and Fig. 1 shows the improved implements.

In the paper referred to I described a gold-silver bead parting-bath, which is the last of the devices used in regular sequence in practice. This bath was designated as being convenient, but was not a labor-saver properly. Fig. 2 of the present paper shows a new device. Instead of the original tray, there are now sectional holders for the test-tubes, each having a wooden handle on either end, so that the holders may be removed from the boiling bath, and the acid or water poured off from each set of tubes (in this case, 7) without waiting to cool. The tubes, held in place by clutches, as shown in Fig. 3, rest in holes in the base-strip, having a smaller diameter than the tubes. Each holder is stamped at either end with a number, so that the bath becomes further useful by permitting several men to use it at the same time without interference. Apart from the handles, the holders are made of sheet-copper.

[1] *Trans.*, xxxvi., 3 to 18 (1906).

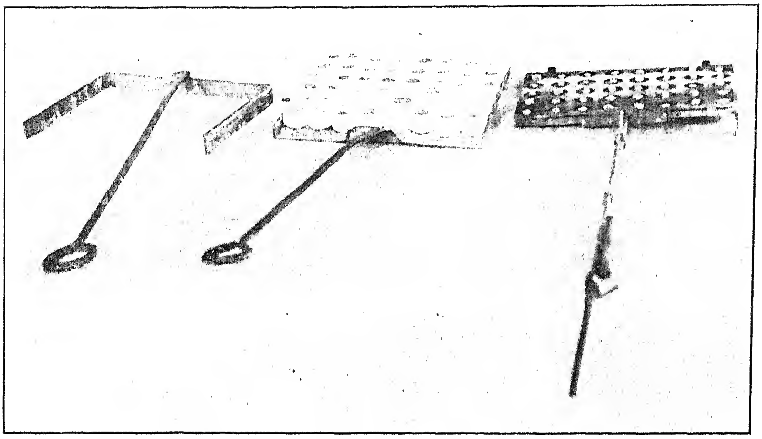

FIG. 1.—DEVICE FOR HANDLING 48 CUPELS AS A UNIT.

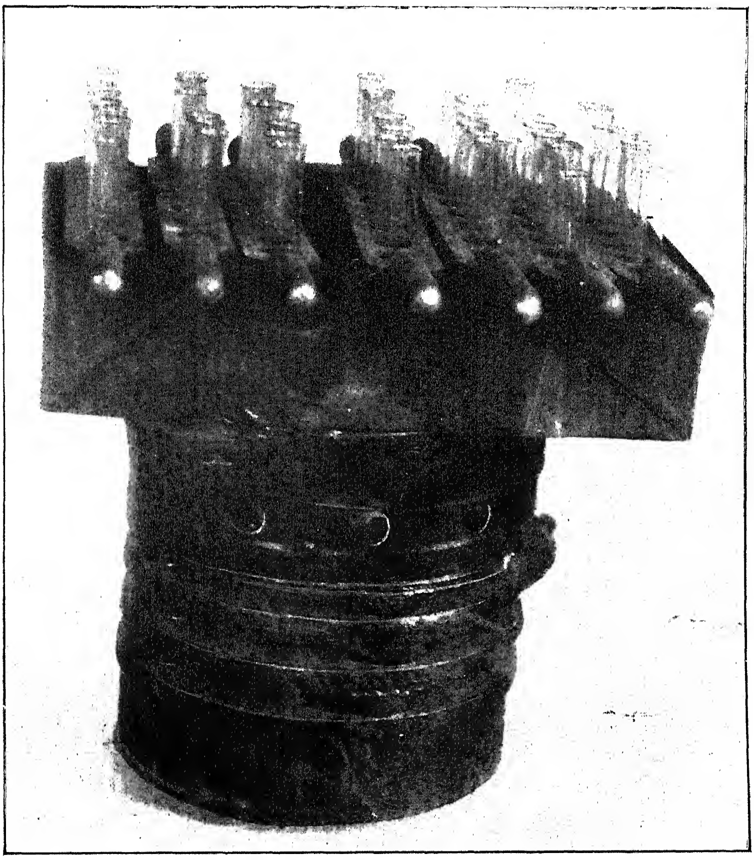

FIG. 2.—PARTING-BATH, WITH SECTIONAL HOLDERS FOR TEST-TUBES.

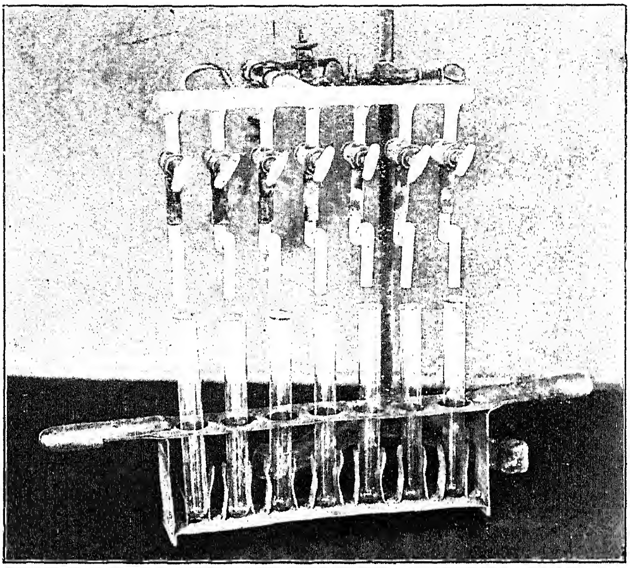

FIG. 3.—TEST-TUBE HOLDER, AND DEVICE FOR FILLING TUBES.

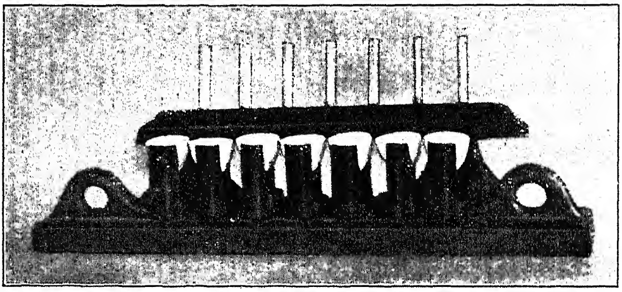

FIG. 4.—ANNEALING-CUP HOLDER.

FIG. 5.—TRAY FOR HOLDING ANNEALING-CUPS.

The temperature of the parting-bath may be raised above the boiling-point of the water by adding to the latter an adequate quantity of glycerine. Salts (sulphates) are not desirable for this purpose, since they have the tendency to creep over the bath.

Fig. 3 shows also the device by which each set of tubes is filled with wash-water. The supply of water is turned on by means of a pinch-cock, and the glass cocks of the individual outlets are set so as to insure an equal stream from each orifice. When in operation, the whole stands in a drain.

Fig. 4 shows an annealing-cup holder, enabling the operator to transfer the gold in sets from the tubes to the cups. The individual clutches are cut from brass pipe. The cup-holder is placed over the tubes in their holder in such a manner that the mouth of each tube slips into a cup. With a little deftness the two holders are then inverted and left together long enough in that position to permit the gold to settle to the bottom of the cups, when the tubes are withdrawn carefully, so as not to spill any of the gold with the water. The glass rods, fastened to a hinged wooden bar, are then turned down over the tops of the cups, and the water from the latter poured off. Were the whole device constructed of metal, the cups with the gold-contents could be dried and the latter annealed therein; instead, they are now transferred to a special sheet-iron tray with a wooden handle, Fig. 5, dried on a hot-plate and annealed on a gas-stove.

The description of labor-saving appliances given in the original and in this supplementary paper demonstrates that, with the sole exception of weighing, the idea of "working in sets" can be carried through every operation in the assay-laboratory and furnace-room.

Mine-Survey Notes.

BY GEORGE W. RITER, SALT LAKE CITY, UTAH.

(Canal Zone Meeting, November, 1910.)

A DISTINGUISHED engineer, the active head of a large mining company, has said that surveying attains the dignity of a profession only in the hands of a few men—the few who know how to keep notes that will be understood by even a novice without further word of explanation.

Of all mine-records, survey notes are perhaps the most important. The maps made from them serve as a working-guide, and as framework for other data; and when, from any cause, access to underground workings is shut off, the surveyor's record often becomes invaluable.

So seldom does a surveyor have a chance to check underground surveys by making a closure, that he is compelled to rely on the precision of each step of his work for the accuracy of the final result. When data are fully recorded on the spot, the notes, even in the hands of second parties, become evidence of the care and precision with which the surveying was done. It is worth a great deal to be able to pick up the work of a former surveyor where it was left off, and to be able to carry it forward without retracing from the beginning.

The purpose of this paper is to describe the survey-note system of a mine now under my direction, in which underground surveys had to be relied upon entirely for the proper division of rich ore-bodies between neighbors, thereby placing a premium on accurate surveying and permanent records.

In survey-work, it soon turns out that paper is the least item of expense, and that little is gained by trying to economize note-book space. For many purposes it is best to record notes on cards or loose-leaves, which may be left in the office at the end of the day's work, and need not be carried into the mine again, there to become smeared with grime and grease. With the loose-leaf or card system, all notes relating to the same local-

ity may be filed or bound together, and additional loose-leaves, containing diagrams or office-calculations, may be inserted opposite the original note pages. If for any purpose a bound book is to be preferred, the books can be so put together that pages containing diagrams, etc., may be interpolated wherever desired and securely fastened. In any event, it is well to keep the field-notes and calculations in such a way as to show which were made in the field and which in the office.

Nothing is more annoying than to discover, while working up notes in the office after the field-work is done, that some important reading has not been recorded and that a special trip to the workings is necessary to get the missing data. In the surveys herein referred to, it was decided to provide blank notes, ruled and printed specially for the work. Each space called for an entry by the surveyor, or else a check-mark indicating that no entry was necessary. With this arrangement, the surveyor could tell at a glance, before picking up his instrument at any station, whether anything had been omitted from his record. Although intended primarily for underground work, the blanks are equally suitable for open-air surveying.

Actual Field-Notes.

Form 1 is the blank for actual field-notes. Each of the three sections into which the page is divided by double horizontal lines, is for the data pertaining to a complete surveyed line. The station occupied by the instrument is recorded at the bottom of the section, and the true course and distance to the next station is recorded in the same section, between the double vertical lines, reading up. If the back-sight does not already show on the page as a preceding surveyed line, it is entered on the same horizontal line as the instrument-station, the arrow, $\overset{BS}{\underset{CC}{\rightarrow}}$, or $\overset{BS}{\underset{CC}{\leftarrow}}$, being placed so as to indicate whether the calculated course, *CC*, runs from the back station, *BS*, towards the instrument, or *vice versa.*

Fr. Mag. and *To Mag.* show the magnetic courses before and after turning the angle at any station. In all cases, *Fr. Mag.* is the magnetic direction from which the angle is turned.

Angle and *Double A.* The plate-angle, read from the ver-

Survey of .....

Date . . . 191. Assisted by . . . ...

READ UP

To

Z (—) R (—) R. (+)

V. ( ) F (+) F (—)

B. S.
C. C.

To

Z (—) R. (—) R (+)

V. ( ) F (+) F. (—)

B. S.
C. C

To

Z. (—) R. (—) R. (+)

V. ( ) F. (+) F (—)

B. S.
C. C.

Fr. Mag	To Mag.	Angle.	Double A.	Slope A. C.	Slope A.	Slp. Dis.	H. I. to	H. O. to	Course Station	Dist	REMARKS.

FORM 1.—BLANK FOR ACTUAL FIELD-NOTES.

nier, is recorded with a prefix, *R*, meaning to the right, or to the left. Compare this with the angle indicated by the ma netic courses. As a check, double the plate-angle.

Slope A. C. is intended as a check on the slope-angle, and the reading that would appear if the graduations on the v tical arc were counted in the wrong direction from the near 10°-mark. For example, if the slope-angle is 18° 46′, then t *Slope A. C.* would appear 21° 14′. Record both readings; t sum of the two is always a multiple of 20°, and the true ang is always the least of the two readings.

Slp. Dis., the slope-distance, is measured from the axis of t instrument, to *O*, the object sighted at. For stadia-wo record here the upper and lower rod-readings; record t middle rod-reading under *H. O. to F.*

Z is a quantity to be subtracted from the slope-distance obtain the horizontal distance. It is the slope-distance mu plied by the versed-sine of the slope-angle. The versed-sine ordinary slope-angles is a quantity having its first significa digit in the second or third decimal place; and with a sli rule graduated for versed-sines, the horizontal distance may determined quite as accurately and more quickly than by m tiplying with cosines.

V represents the vertical distance between the axis of the strument and the object, *O*, sighted at. It is the slope-dista multiplied by the sine of the slope-angle. In most cases *V* m also be found with sufficient accuracy by using the surveyo slide-rule, graduated for sines, tangents, and versed-sines.

H. I. to is for the height of instrument above the floor, *F*, to the roof, *R*, as the case may be in mine-surveying.

H. O. to is for the height of object sighted at, measured fr the floor, *F*, or to the roof, *R*. For stakes and points in surfa work, enter *H. O. to F.*

Remarks.—The description of each permanent survey-poin entered opposite the station number. The distances to obje noted along the line may be entered in this column as *plus* and are to be measured from the instrument-station in each ca All recorded distances to the right or left are understood to measured from the surveyed line at right angles, unless oth wise stated. Whenever the space in the "Remarks" colu

Office Record of Survey . ...

Surveyed .. .. .191. Calculated.. .. . . 191 ..

READ UP

Course · Dist. Station	Latitudes N (+) S (—)	Departures E (+) W (—)	ELEVATIONS. Floor Point (F) Roof Plug (R)	Log. Cos.	Log. Dis.	N.(+)S.(—)	Log. Sin.	Log. Dis.	E (+)W.(—)

FORM 2.—BLANK FOR OFFICE RECORD.

is not ample for sketches, use reverse side of sheet or else a second sheet for this purpose.

Office Calculations.

Form 2 is the blank for office calculations, and if so desired a carbon manifold copy may be made to serve as a duplicate office record. The last two columns are intended for calculations by logarithms. The latitudes, departures, and elevations are recorded opposite the stations to which they belong.

In the surveys herein referred to, all boundary corners were tied to the nearest U. S. land monument, and the origin of latitudes and departures was so chosen that this monument became Latitude N. 5,000 and Departure E. 5,000. By this means, none but positive quantities were needed to express the latitude and departure of any point in the neighborhood.

An underground closure sometimes calls for slight change in the meridian as first carried underground, in which case the latitudes and departures should be re-calculated for complete accuracy. With the loose-leaf system, new sheets containing the modified calculations may be interpolated without confusion.

Maps.

All original maps are made and kept up-to-date in the main office; and being made on indestructible linen, map and tracing are combined in one. Blue-prints on linen are made from the original map for the use of mine-superintendents or foremen, who are free to make additional marks thereon for their own purposes. When a new set of blue-prints is made, the old set is taken up and filed away.

Perpetuating Survey-Points.

For perpetuating survey-points, brass screw-eyes, brass nails, and brass tags are used, on account of being non-corrosive. Points are always set overhead if the ground is firm enough. A turned wooden plug, previously soaked in some wood-preservative, is driven into a drill-hole until solid; then the end is daubed over with bright red paint to make it conspicuous; after which a screw-eye, threaded through a numbered brass tag about as large as a silver dollar, is set in the bottom of the plug. If the roof is insecure, a screw-eye is put in a mine-

timber overhead, or a nail is driven into a cross-tie underfoot; but the screw-eye or nail always goes through the middle of a numbered tag so as to hold the tag against the wood; this prevents loss of the identifying tag so long as the screw-eye or nail itself remains in place.

The numbers from 100 to 199 were assigned to the first level, from 200 to 299 to the second level—and so on, 100 numbers to each mine-level. Certain numbers out of each hundred were assigned, in turn, for use in one general direction from the main shaft, and the rest for use in the opposite direction. Thereafter, all stopes, chutes, etc., were designated by the number of the nearest survey-station, and all assay-samples, etc., were recorded with similar reference. A sign-board at each ore-chute, marked in large figures with the number of the nearest survey-station, made things easier for the bosses and workmen. For example, the mark "636" on a stick of timber delivered at the main shaft would constitute full directions for its delivery on the sixth level, north side, at the station named.

After being in use for more than 15 years, during which period the underground workings of the mine have grown to an aggregate length of more than 25 miles, this simple system of numbering is still satisfying all requirements. To some extent, the feasibility of such a simple system of numbering has hinged on the development of the mine through one main shaft, with levels at regular intervals; while irregular and more varied workings might have called for something more elastic. Wilbur E. Saunders has described A Reference-Scheme for Mine-Workings[1] which has unlimited elasticity as a feature. Between the two extremes, the range is sufficient to satisfy almost any special condition.

Although this paper deals with an elementary subject, I believe that it will be appreciated by every one who has ever had to deal with a set of survey-notes that were worth but little more, in the absence of their author, than the paper on which they were written.

I desire to give to George W. Snow, civil engineer, Salt Lake City, Utah, much of the credit for the system described in the foregoing paper.

[1] *Trans.*, xxxvii., 128 to 139 (1907).

Dry-Washing for Placer-Gold in Sonora, Mexico.

BY J. V. RICHARDS, SPOKANE, WASH.

(Canal Zone Meeting, November, 1910.)

THE Altar district, State of Sonora, Mexico, is for the most part a desert with but little rain-fall and few running streams. On account of this scarcity of water it is necessary for the natives to "dry-wash" the placer-gravel, and the object of the present paper is to describe some of the more common Mexican dry-washing devices.

My observations were confined to a property known as Las Palomas placers, 15 miles from the Gulf of California and about 150 miles south of the United States border; a brief description of which property may be of interest as illustrating what is, perhaps, a typical Sonora dry placer.

It is said that as long as a century ago the Yaqui Indians discovered rich cement-gravel, known as *argo masa*, at Las Palomas, and that there has been a more or less steady production of gold from the property ever since. The *argo masa* is a fairly-coarse gravel, the pebbles being quite firmly held together with a lime cement. Gold is scattered through this cement in particles ranging from fine specks to pieces several grams in weight, and occasional large nuggets are encountered. The contents vary so widely that it is difficult to estimate the average value, but returns of from $10 to $50 per cubic yard are common, and it is doubtful if the Indians considered anything less than from $4 to $5 a yard as "pay dirt." This *argo masa* occurs in channels varying in thickness from a few inches to several feet, and in width up to 300 ft. These channels are continuous and are probably ancient stream-beds. They are for the most part covered with an over-burden of a more recent gravel, varying from 30 to 50 ft. thick, and the fact that this over-burden contains values as high as $1.50 per cubic yard, and yet was not worked by the Indians, shows that their idea of what constituted pay-ground was high.

The old Yaqui method of working this ground was to sink a shaft through the over-burden and gouge out the *argo masa*, as far as possible, from the bottom of the shaft. As they had no timber to support the roof, cave-ins were frequent and only a limited area could be worked from one shaft. For this reason the shafts are very numerous, and often are not more than 30 ft. apart.

The cement is hoisted to the surface in small rawhide *tenates*, and women and children pound it up between boulders. The coarse gravel is removed and the pulverized cement panned in large wooden *bateas*, shown in Fig. 1. Both the Yaquis and Mexicans are wonderfully expert at this sort of work and little, if any, of the coarse gold is lost. The method of operation is, however, exceedingly slow and laborious, and with the invention of the Mexican dry-washer more material could be handled, though the preparation of the cement for washing was still the same.

The dry-washer is an air-blast machine, Figs. 2 and 3, working on the difference in specific gravity of metal and gangue. It is really a small table with a cloth top and a bellows below. It is about 4 ft. long by 2 ft. wide, and has a hopper at one end through which the gravel is fed. The top of the table consists of a fine wire netting like ordinary fly-screen. On this is placed one and sometimes two layers of burlap, and over this a layer of thin cotton cloth; old flour-sacks are often used. Four 0.5-in. riffles are fastened at equal intervals across the surface. Detachable wooden sides are used to keep the gravel from spilling over. The machine is usually built to stand level, but is set on the edge of a dump at an angle of about 20° from the horizontal. It is kept carefully leveled in cross section.

A hand-crank belted to a small drive-wheel operates the bellows, which has the ordinary clap-valves in the bottom and canvas sides. The intermittent puffs of air from this bellows, coming through the cloth top at the rate of about 150 per minute, cause the sand and gravel to jump the riffles and travel down the table and over the end. Most of the gold, together with a considerable amount of black sand, is caught behind the first two riffles. The capacity of such a machine is from 2 to 2.5 cu. yd. per hour.

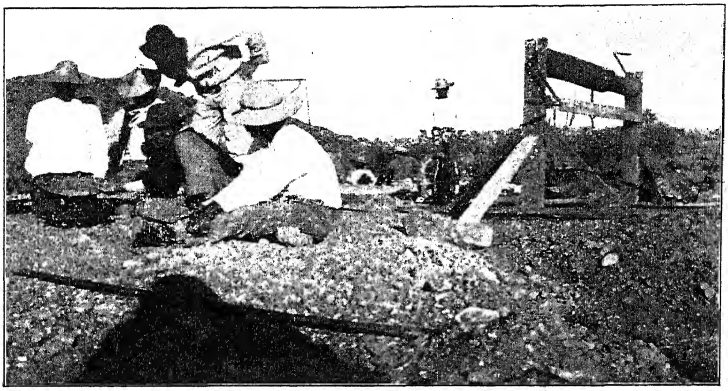

FIG. 1.—DRY-PANNING IN BATEA, LAS PALOMAS PLACER, SONORA, MEXICO.

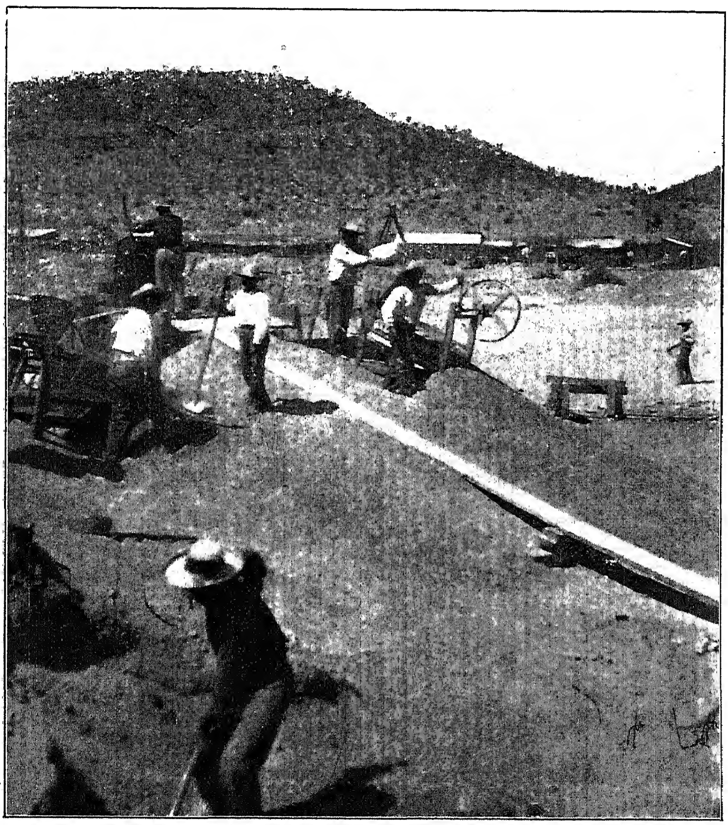

FIG. 2.—PRODUCT FROM DISINTEGRATOR BEING PUT OVER DRY-

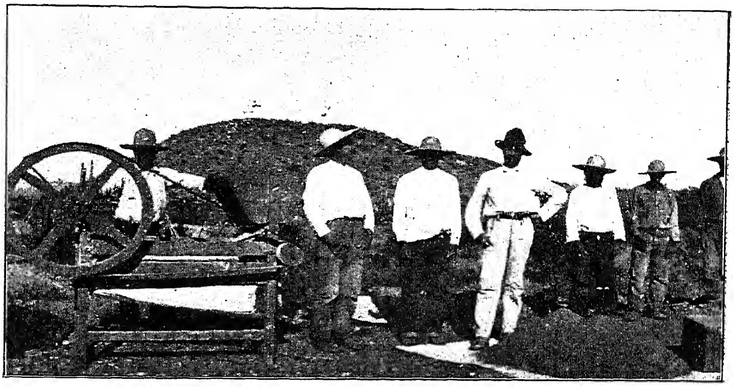

FIG. 3.—MEXICAN DRY-WASHER, LAS PALOMAS PLACER, SONORA, MEXICO.

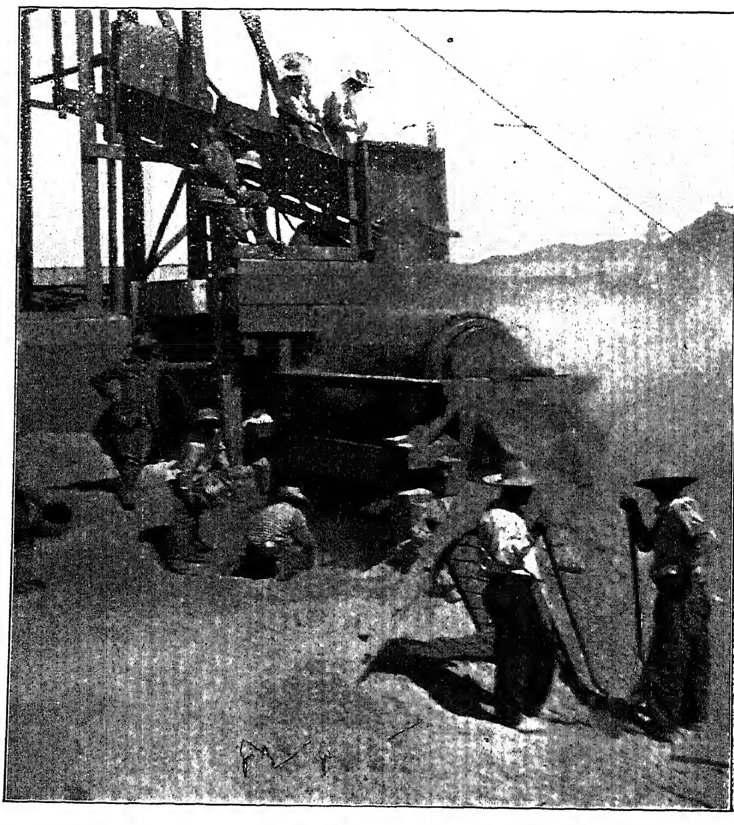

FIG. 4.—QUENNER DISINTEGRATOR IN ACTION, POLUDÀ, SONORA, MEXICO.

After several yards of gravel have been passed over the table, the machine is stopped, the side-boards removed, and the material behind the riffles brushed up into a gold-pan. It is then fed back slowly, nearly all the gold this time remaining behind the first riffle. The first and second riffles are then brushed out clean and the resulting sand, black sand, and gold is tossed in a *batea* and the gold separated dry and clean.

For the successful operation of the dry-washer, it is essential that the sand be absolutely dry. Even a small amount of dampness tends to clog the cloth and the riffles. In the case of a cement-gravel it is also, of course, necessary that the material fed to the machine be thoroughly pulverized. Hand pulverizing being exceedingly tedious, several disintegrating-machines have been devised for this work. Most of these machines are faulty in the respect that they not only pulverize the cement but smash up the pebbles as well, which is obviously useless labor, as all that is required is to clean and eliminate the gravel and pulverize the cement. A machine known as the Quenner disintegrator (named after the inventor) does this work more or less successfully. It consists, Fig. 4, of a cylinder 7 ft. long and 4 ft. in diameter, made up of 1.5-in. steel strips with slits about $\frac{1}{8}$ in. between them. In the center is a shaft, to which are attached short chains having tough, heavy manganese-steel lugs, the size of a fist, at the free ends. The cylinder revolves in one direction at about 40 rev. per min., while the center shaft revolves in the opposite direction at four or five times this speed. The cement gravel, fed in through a hopper, is thoroughly broken up by the action of the chains and lugs, the fine material dropping out through the slits, and the coarse pebbles being ejected through the end of the cylinder.

At the camp of Boluda there are three Quennor disintegrators, operated by gasoline-engines, which are said to turn out a satisfactory product for dry-washer treatment.

At first sight one might be led to suppose that this type of dry-washer is somewhat crude and inefficient, but I took great pains to arrive at an idea of the percentage of gold lost in the tailings, and while accurate results could not be obtained on account of lack of appliances, the amount of gold recovered from the tailings was very small.

The test was as follows: 27 cu. ft. of loose over-burden gravel, which did not require disintegrating, measured with as much accuracy as possible in a wooden box containing exactly 1 cu. ft., was screened and passed over the machine. The clean-up from the riffles was panned in a regular gold-pan in water and the tailings from the first pan caught in a second and re-panned; the gold was collected with mercury, separated by treatment with nitric acid and finally dried and weighed. The result showed a value of 70 cents per cubic yard, estimating the value of the gold at $15 per ounce.

The dry-washer tailings were then put over the machine a second time, and as the material fell from the end of the table it was sampled as carefully as possible, one gold-pan full being intended to represent an average of the tailings. The weighed clean-up from the riffles gave a value of 4 cents. The pan of tailings, on being twice panned in water and amalgamated, gave an amount of gold almost too small to weigh on the assay-balance available. Its value was about 0.5 cent; and as there were 12 cu. ft. of tailings (the remaining 15 cu. ft. being coarse pebbles previously screened out) this value represented a total value of 6 cents for the tailings. To this amount add 4 cents recovered from the riffles and the total loss was 10 cents, or an apparent extraction in the first operation of 87.5 per cent. Doubtless considerable fine gold was lost in the panning and also blown from the machine, but it seems that with fairly-coarse gold, such as that at Las Palomas, the percentage of recovery is in excess of 80 per cent.

The Solid Non-Metallic Impurities in Steel (Sonims).

BY HENRY D HIBBARD, PLAINFIELD, N. J.

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I. Introduction.

THESE impurities are perhaps the most important things in steel—especially steel made by the oxidation processes—the effect of which has not been at least approximately determined. By oxidation processes I mean the Bessemer and Siemens-Martin, during which the metalloids in the charge are removed by oxidation.

It has been truly said that it is a clear advantage to have a new name for a new thing. To secure this and to economize space, let us adopt the name *sonim* (plural *sonims*) to designate the solid non-metallic impurities existing in steel or other metal. I would define a sonim as a solid non-metallic portion of matter existing as an impurity in metal. A piece of sand, brick, clay, or such material, imbedded in metal, would be considered as a foreign body, not as an impurity, and therefore not a sonim.

The studies of sonims of which the results have been published hitherto, have been made chiefly or wholly on finished or at least on cold steel, without much, if any, regard to the specific methods by which the steel was made. While many of the results so obtained are valuable, they give little help in determining the cause, effect, or cure of the disease. It is not enough, in such a study, to know merely the commercial name of the process; for within the field of each process lies a great number of possible variations, each of which will yield a result differing in some respect from all others; and the control of sonims is particularly dependent on the metallurgical treatment of the metal, because of the inability of analytical chemistry to throw much light on their presence, condition, and effect, as they usually occur in commercial steels. The microscope aids more, perhaps, than chemistry in their study.

When collected into large globules, the sonims may be seen, with the use of a good magnifying-glass or even by the unaided eye, on a polished section of steel which contains them. Smaller ones may be seen with the aid of the microscope. The quantitative determination of their composition, amount, manner of distribution in the steel, and effect on its properties, presents most complex problems which can never be solved when starting with the finished steel. The study must begin not later than with the steel-making operations.

While it is thought worth while to study the sonims separately, as distinct from the metallic as well as the gaseous impurities, the whole question of non-ferrous matter in steel is in a certain sense actually involved, since, in addition to its mere quantity or percentage, the chemical condition of such matter, that is, the chemical compounds it forms, wholly or in part, ought to be determined, and also the manner of distribution of such chemical substances throughout the steel, if the whole is to be understood. This is the aspect of the case which makes an ultimate analysis, which gives only the percentage of each element present, often an insufficient explanation of the peculiarities of steel, even though the analysis itself be complete and accurate. The analysis is one thing, but its correct interpretation is quite another; and the realization of its limits as a source of knowledge is still another.

The metallic impurities, it may be assumed, are actually in a state of solution in steel. The gaseous are in solution until they separate as gas. The sonims are sometimes clearly insoluble in steel, but in other cases are so finely disseminated in the metal that it cannot be said whether or not they are in solution. Iron oxide is commonly asserted to be dissolved in the metal.

What the chemist reports as silicon in steel may have existed there in iron silicide, which, in well-made steel, made by the solution method, does good by favoring solidity; or it may have been in the form of oxide, or as one of the silicates noted below. The presence of 0.01 per cent. of Si may then mean about 0.02 per cent. SiO_2 or 0.10 per cent. of $3\ FeO,SiO_2$. This silicate, having a specific gravity about half of that of iron, will represent sonims of 0.2 per cent. of the volume of the steel. This amount, or even less, when existing in certain ways considered

below, may account for almost any degree of poor quality i the steel.

Any oxygen in the steel (which the chemist does not report may exist in any of the sonims containing SiO_2, or it may b there in Fe_3O_4, FeO, MnO, or possibly in some other oxide In the open-hearth process, iron oxide is taken up or dis solved from the irony cinder formed in melting, or resultin from the ore added to the bath. SiO_2 and MnO are forme within the metal by reactions of the iron oxide with Si an Mn. In the converter, oxides are formed in the metal by th O of the blast.

In molten decarbonized iron prepared in the converter o open-hearth furnace, there may be existing in the solid stat any or all of the sonims SiO_2, Fe_3O_4, and MnO; in the fuse state, any or all of the silicates of iron and manganese.

In the molten steel ready for teeming, as in the ladle, ther may be the same sonims as in the decarbonized iron, but i very different proportions and conditions. They should al have become liquid and have been largely eliminated; whil the gaseous impurities present should be held more securely i solution than before the final additions.

The solidified steel in the ingot or casting should contai still less of the sonims than that in the ladle. Such gases a have not been worked out or kept in solution will exist i blow-holes in the steel. We will not consider them further.

A complete list of the sonims in steel may include FeO Fe_3O_4, MnO, SiO_2, FeO,SiO_2, $2\,FeO,SiO_2$, $3\,FeO,SiO_2$, $4\,FeO$ SiO_2, MnO,SiO_2, $2\,MnO,SiO_2$, $3\,MnO,SiO_2$, $4\,MnO,SiO_2$, MnS FeS, P_2O_5, and possibly calcium silicate. If other element have been added, such as Al, W, Cr, Ti, or V, their oxides an silicates may be present.

Any of these sonims and perhaps others (some of whic have been suggested by writers on the subject) may be pres ent, and mixed in almost any proportions, forming any on of a very large number of possible mixtures or combinations The analyses hereafter given indicate that they are, in som instances at least, largely sub-silicates, with three or fou molecules of base to one of silica.

Phosphide and silicide of iron existing in the steel may b considered as metallic impurities outside our subject. Whe

present in small quantities, they raise the tensile strength of steel, acting more like iron carbide than like sonims. It is claimed, and may be true, that sometimes iron oxide also raises the tensile strength of steel; but in some of its forms it probably has the opposite effect: and it is doubtless proper to class it as a sonim, or non-metallic impurity.

II. An Illustrative Experiment.

To illustrate by an extreme example the overwhelming effect of sonims, chiefly SiO_2, in steel, the following account of an experiment will serve.

The point had been raised that perhaps straight silicon steel would be found to have desirable properties, and that Si might prove to be as efficacious in removing O from molten steel as Mn. To see what there was in this idea, I made a heat of steel in a 12-ton open-hearth furnace and added ferro-silicon at the end, instead of ferro-manganese or spiegel. This was many years ago, and I have not now the analysis of the steel; but C and Si were each about 0.3 per cent. The heat was made by the pig-and-scrap method, no ore being added. I melted it myself. The fuel was producer-gas. The heat was normal until the addition of the ferro-silicon at the end, when the boiling ceased and the metal was cast into 12- by 12-in. ingots, into which it flowed quietly like milk. It lay dead in the molds and solidified without action, except to settle at the top, forming a pipe as freezing proceeded. So far all looked well.

The ingots were reheated and one of them received one pass through the blooming-mill. The result was striking. The ingot was reduced almost to sand and gravel. It was red-short to a most astonishing degree, breaking into a myriad of pieces, quarts if not pecks of which fell through between the rollers of the blooming-mill tables. The thousandth part of the red-shortness possessed by the steel would have made it valueless for any purpose for which steel is required.

Afterwards, I recalled a statement made in Holley's Report to the Bessemer Steel Association, to the effect that somewhere in Europe (Belgium, I think) he had been told that they considered one function of the manganese added was to remove silica from the steel. The experiment demonstrated pretty

clearly the truth of the statement. It seemed evident that the red-shortness of this steel was largely due to the reaction of the added Si with the O in the metal, forming SiO_2, which floated in the liquid steel as so much foreign matter, breaking everywhere the continuity of the metal, which, as a result, could not be hot-worked in the slightest degree without rupture.

No examination was made of the physical properties of the metal, cold, the ingots being broken up and remelted. Metallography was not then in use, but a microscopic examination of that steel might now be instructive, could it be made. Similar steel could, of course, be made and examined by any one desirous of studying an extreme case of the disease. The SiO_2 had apparently little or no power to unite with the iron oxide to form a fusible silicate. It did not seem likely that the metal, if cast without any final additions at all, would have been nearly as red-short as it was.

The Si added did, however, have a pronounced effect on the solidity of the ingots, apparently as much as if added with the usual amount of Mn. Whether or not there was any reduction of occluded CO by the Si cannot be told; but, either by that action or by retaining the gases in solution, the formation of blow-holes was prevented. If by the latter action, then it argues that all the Si was not oxidized, but that a considerable part remained as silicide, increasing the solvent power of the steel for gas, but showing a proportionately greater power for harm for each unit of SiO_2 in the steel.

As bearing on the existence and effect of SiO_2 in open-hearth steel, the experience next described has value and it was very important at the time it occurred.

After having had some experience at melting steel by the pig-and-scrap method, in which no ore was added to the bath, I was obliged, at another place, to make steel by the excess-of-pig-and-scrap method, the excess of Si and C being worked out with ore, a practice which requires methods quite different from those presented by the pig-and-scrap method proper. I was melting soft steel for boiler- and ship-plate, and the plates made were passing inspection with some success; but they showed a tendency to red-shortness, and some had "snakes" on the surface, while the elongation in the pulling-test was usually close to the lower limit permissible when the plates

were sold on specifications. The only thing unusual in the analysis was high Si, ranging sometimes as high as 0.02 or 0.03 per cent.

Concluding, perhaps naturally, that the Si was the cause of the trouble, I tried to eliminate it more completely from the metal of the next heats by oxidizing it with more plentiful additions of iron-ore. To my surprise the chemist reported higher Si than before; and the quality of the steel, as developed in rolling and testing, was worse instead of better. As usual in such cases, I questioned the accuracy of the analysis; but the chemist stood his ground, and events proved that he was right.

The next procedure was, of course, to go in the opposite direction: to use ore less freely and to allow time at the end for some of the iron oxide in the slag to be worked out by reduction and combination with SiO_2 of the acid hearth, the sonims in the metal being worked out at the same time. The oxides in the slag and metal followed similar courses as though there were some balance between them, that in the metal varying roughly with that in the slag.

The quality of the steel was improved by this procedure; and, on our going the same way to the reasonable limit, all signs of red-shortness and "snakes" disappeared, and the plates scaled cleanly and freely in rolling, which they had not done before, even when they were not pitted (a defect due to other causes). The elongation in the pulling-test went up also.

It thus became clear that, so long as there was Mn remaining in the metal, it was continually being oxidized, and the resulting MnO united with the SiO_2 as it formed, making a fusible silicate which collected together and floated out, collecting other oxides as it went, cleaning the metal, and eventually joining the slag. But after the Mn was all gone the SiO_2 formed floated in the bath unfused and therefore without power to collect into particles large enough to float on the top. Hence, when the oxidizing-conditions were intensified by larger additions of ore, the Mn was eliminated earlier than before, and the SiO_2 remained in larger quantity in the metal. In other words, the elimination of SiO_2 was seriously checked as soon as there was no MnO to flux it within the mass of the metal; and it remained suspended as so much foreign matter in the steel.

When I had previously made steel by the pig-and-scrap method, there was considerable Mn in the scrap added; and, as the charge was merely melted, and tapped as soon as the final additions were melted, there was always some Mn from the initial charge persisting to the end, which kept the SiO_2 in the steel within reasonable limits, without our making any particular effort to that end. In that former practice I had improved quality by allowing, after melting was complete and before putting in the Mn, a little more time than had been customary at the plant; the result being that the ingots developed fewer cracks in the early passes in the slabbing- and blooming-mills, undoubtedly because of more complete removal of the sonims.

III. Scum on Ingots.

When soft steel of a commercial quality, made by what I have elsewhere styled the "evolution" method, in which the gases are encouraged to evolve freely during solidification, is in the molds, there is a very active escape of myriads of small gas-bubbles, which, in rising, stir the metal very thoroughly, until it freezes or becomes pasty. During this evolution of gas a non-metallic slag-like scum, more fusible than the metal, collects on the top of the ingot. In the acid open-hearth process it is sometimes olive-green or gray, though it may be black; and it is sometimes crystalline, though usually vitreous. On basic ingots it is black and crystalline.

Table I. gives analyses of some of these scums, which show clearly that they are not a part of the slag. These samples were vitiated by dirt, clay, and refractory materials, which got into the molds in various ways and were dissolved by the scum; and, as a consequence, the SiO_2 and Al_2O_3 are too high. The quantity of this scum may be seen to increase after teeming has ceased, so that further additions to it can only come from the metal. The final amount may be from 4 to 8 oz. per ton of steel or even more. This substance is, without doubt, chiefly sonims remaining in the steel when teemed. Their normal behavior in practice is considered elsewhere.

TABLE I.—*Analyses of Scum of Sonims from Ingot-Tops.*

	Acid Bessemer Pipe-Steel, First Ingot.	Acid Open-Hearth Boiler-Steel.	Basic Open-Hearth Boiler-Steel.
	Per Cent.	Per Cent.	Per Cent.
Iron (Fe)	....	15.96	15.35
Silica (SiO_2)	7 95	20.38	17.14
Phosphorus	0.043	trace	0.03
Manganese	38.66	48.50	36.00
Alumina	....	4.00	15.08
Sulphur,	2.51	0.39	0.16
Lime (CaO)	......	......	2.04

The elimination of phosphorus and particularly of sulphur from acid steel shown in these analyses is interesting.

Scum sometimes collects on the tops of high-carbon ingots —those which do not pipe, but the metal of which is kept in some motion by escaping bubbles.

When drillings taken from a test-piece of an unfinished decarbonized open-hearth charge are being dissolved in 1.2 sp. gr. nitric acid, close observation will discover small quantities of minute black particles, separating from the dissolving metal. After a long series of observations, in which the charges known to be over-oxidized showed more of these particles than others, I came to consider them as iron oxide (Fe_3O_4) and to judge from their quantity of the degree of oxidation of the metal.

Rail-steel ingots examined within the past year or two at Watertown Arsenal, showed numerous sonim globules within 1 in. of the bottom of the ingot, and far fewer, perhaps not over a fifth as many, 6 in. from the bottom, where the steel had remained liquid about 15 min. longer. It was Bessemer steel made by a quasi-solution method, the ingots showing many blow-holes; but there must have been little or no agitation after the steel had become quiet in the molds, and the relative freedom from sonims of the metal 6 in. from the bottom surface must have been due to the period during which the metal was still liquid, and the sonim globules were floating upward, to collect in or about the pipe and central segregation, adding to the injury from those defects. These globules must have largely been formed in the steel before it was run into the molds and the whole mass must have contained sonim globules as plentifully as the metal in the outer skin of the in-

gots. The sonim globules, while very numerous, were not absolutely ruinous to the steel; and their effect in the interior, away from the skin, where their quantity was greatly reduced, might be considered almost negligible, except in the upper central part of the ingot where they collected.

IV. The Effects of Sonims.

Until much more is known about sonims and the way in which they exist in steel, little can be said quantitatively of their effect on the finished product.

From the experience previously described, with the excess-of-pig method, I concluded that when soft open-hearth steel made by the evolution method, and to which no Si had been added at the end beyond the small amount in the ferro-manganese, contained more than 0.01 per cent. of Si, there was enough SiO_2 present to injure commercially the quality of the steel. It may now be said that 0.01 per cent. of Si in the form of sonims may be so distributed in soft steel as to make it noticeably red-short, and to lower the elongation one-tenth, or say from 28 to 25 per cent. in 8 in. That amount may also be distributed so as to have a far greater effect, as in the experimental heat described.

The effect of the sonims in steel is practically never proportional to their quantity. Much depends on the way in which they exist. In relatively large globules, the effect, unless their number be abnormally large, is not very important in the commoner grades of steel. An exception to the latter statement is sometimes presented when an extraordinarily large globule or group of globules has formed, which for some reason has not floated out of the metal, and which may, when extended by forging or rolling, form a weak spot or surface in the resulting article, which is a flaw, and may lead to rupture in service. Apart from such exceptional globules, which may reach $\frac{1}{16}$ in. in diameter, sonim globules are rarely more than 0.01 in. in diameter, and may exist in any size from that down to single molecules, of which millions are required to make a globule 0.01 in. thick.

Both chemistry and microscopy may fail to determine the presence and amount of sonims in finished steel, when they are in their smallest and most hurtful forms. The mysterious

lack of quality of certain steels may perhaps be explained by the presence of sonims, particularly steels made by the oxidation (Bessemer and open-hearth) processes. It has been common to ascribe chiefly or solely to the presence of oxygen in the steel such of their effects as have been recognized. With the control and elimination of sonims to the proper degree, steel-making will make a long step towards being an exact art.

Discrepancies are continually found between the physical properties of steels of substantially identical composition, made by the same process, the same plant, and the same men, and commercially of good quality. Differences occur in the properties of steels due to the different processes by which they were made, and abnormal heats are met with having properties which do not nearly coincide with what would be expected from their analyses.

We are accustomed to ascribe such variations partly to the unavoidable inaccuracy of our determination of composition by the analyses of samples or drillings; partly to segregation; and partly to the temperature and other features of mechanical treatment. To this list of possible causes, I think one more may be added—namely, that of the presence, condition, and distribution of the sonims at the time of solidification.

To put the matter in another way: There are certain perfect physical properties which belong to a perfect piece of steel (which, strictly speaking, cannot be made) of any given composition, as the term "composition" is usually understood; but in practice each piece of steel of that analysis falls short of those perfect properties to some extent, according as it is better or worse made. All steels have defects in some degree, and one of the causes is often the presence of sonims.

To consider the effect of sonims as affected by their size, let us take a piece of steel containing a sonim globule 0.01 in. in diameter to each square inch of cross-section, the steel having a normal tensile strength of 60,000 lb. per sq. in. The globule in question will have an area of cross-section of about $\frac{1}{12,000}$ part of the square inch, and, by displacing that amount of steel, will reduce the tensile strength about 5 lb. per sq. in.—a wholly inconsiderable amount. A hundred such globules per square inch would decrease the tensile strength only about 500 lb. per

sq. in. Therefore a few sonim globules of that size may have a negligible effect.

What the sonims represented by the 0.01 per cent. of Si, which seriously affected the physical properties of the ship-plate above referred to, were, and how they were distributed in the steel, is, of course, a matter for conjecture; but it seems plausible to conclude that their greater effect may have been due in some degree to their existing in smaller particles and in some particular manner of distribution throughout the steel.

There are two ways in which a given percentage of sonims may have a greater effect when the individual particles are of smaller size. One is, that the smaller particles, if globes or other regular geometrical figures, will have a greater aggregate area of cross-section per unit of volume or weight than larger particles, according to the geometrical rule that the volumes of similar solids are proportional to the cubes while the areas of their cross-sections are proportional to the squares of their homologous dimensions. The other and probably more important way is that the sonims when very small may be located largely along the contact-surfaces of the grains formed when the steel solidified, in which positions they would evidently cause weakness and reduce greatly the tensile strength and ductility. Larger sonim globules, which have been under motion due to gravitation, seem to be located at random throughout the steel, except that they are sometimes frozen in the act of collecting into clusters near the center of the mass.

When Si is high, *i. e.*, over 0.01 per cent. in soft steel, it is likely to be largely in the form of oxide or silicate.

A very little sonim on the wall of a blow-hole will prevent the welding-up of that hole in rolling or forging.

Sonims may possibly be one reason for the poor quality of open-hearth and converter steel cast at too high a temperature. These steels in the presence of plenty of iron oxide seem to absorb or dissolve the more of it the hotter they are; and the higher temperature may check the reduction of the iron oxide by the Mn added. If so, the reactions continue after the metal is in the molds, causing gases which form blow-holes, and, what would be far worse, the sonims resulting have no time to escape, but remain in the metal, interrupting its continuity and making it red-short. Probably they would be

locally segregated along the joint-surfaces of the columnar crystals often found in the outer layers with their longer axes normal to the cooling surface, which, if it occurred, would account amply for the red-shortness of steel cast too hot.

Good crucible steel cast at a higher temperature than open-hearth or converter steel is none the worse for it, because, it may be, the steel is finished in the pot and then cast. There are, or should be, no final additions or reactions to be completed while the metal is moving to the mold.

V. The Elimination of Sonims.

The metal when ready for the final addition of Mn, contains oxides and sulphides, floating or suspended, the oxides being infusible at the temperature of the bath, and in extremely fine particles too small for gravity to affect appreciably—much as finely-powdered silica and iron-ore will float in water with which they have been mixed, making it roily or turbid, though with the difference that the steel does not wet such suspended matter. The sonims at this stage have little or no power to flux each other, or to unite and form globules of larger mass.

The effect, on the sonims, of the final addition of Mn to steel containing them is to change their composition and lower their fusion-points, upon which they begin to agglomerate and escape from the metal. So the final addition of Mn, as usually made, washes out a good deal of the sonims; and if the addition be large enough, and sufficient time be allowed after its addition before teeming, the result will be tolerable as to the properties of the product.

Let us first follow the course of the silicates, which are perhaps the most deleterious of the sonims. The MnO formed by the reaction of the Mn added with a part of the Fe_3O_4 (with the formation of FeO) fluxes the SiO_2, forming a fusible silicate, the particles of which wet and flux each other when they touch, and join together, making larger and larger particles. The molten metal and sonims do not "mix" any more than oil and water, nor do the fused sulphides mix with either.

The SiO_2 seems not to be able of itself to split up Fe_3O_4 so as to combine with the FeO resulting. Nor does Fe seem to be able to reduce Fe_3O_4 to FeO, as Mn does. At all events the

iron oxide lacks for some reason the power which the MnO possesses to flux the SiO_2.

When SiO_2 is high in soft steel it is often if not usually accompanied by high Fe_3O_4 while, if they fluxed each other, only one of them—the one in excess—could be high.

The temperature of formation of all the usual silicates of iron given in the books is indeed lower than that of molten steel; and the exact reason why SiO_2 and Fe_3O_4 do not flux each other must be sought elsewhere than in that temperature. The small amounts of SiO_2 and iron oxide may simply be kept apart by the relatively great mass of metal, but that is not reasonable; else why is the MnO formed able to get at and flux the SiO_2? Carbon is also present and should be able to reduce the iron oxide, which indeed it does, but rather slowly and incompletely.

The agglomerating-operation continues until the particles are large enough to be affected by gravity and to rise towards the top of the metal. Each fused sonim globule also wets any other oxide particle which it touches in its travels, fluxes it, and unites with it to form a larger particle.

At the same time, the FeS which floated in the metal unable to agglomerate is attacked by the metallic Mn and fusible MnS is formed, the particles of which act in a way similar to that of the silicates, collecting together by wetting and amalgamating into larger and larger globules, which may exist by themselves or be mixed sometimes, though not intimately, with collections of silicates. Any S in a sonim must, of course, have come from the metal and not from the slag.

Time and moderate agitation of the molten steel are effective in enabling it to rid itself of sonims, provided that they wet and flux each other. If they do not, then agitation will tend to prevent their separation except, usually to a small degree, in the presence of a little Mn in the metal or of an acid slag. Therefore stirring before the final additions, though useful to dislodge dissolved gases, is usually of slight effect in cleaning the metal of sonims. The exceptions to this statement above noted may, of course, be so emphasized as to be sufficiently effective. That is, an amount of Mn which would be small compared with that in the final additions, or a strongly acid vitreous slag, or both together, would be effective—given

enough time, or agitation, or both. The time is required for the reactions to be completed, and, after that, for the sonims formed by those reactions to escape from the metal.

In the low-carbon "evolution" steel which has been considered, the stirring which results from the escape of the gas-bubbles helps greatly the escape of the sonims which are then in a state of fusion and which continue to rise and enter the scum until the metal becomes too pasty or solid. The sonims which escape in the mold come chiefly from the interior parts of the ingot, which have remained molten longer than the metal near the outer surface. The latter congeals before the sonims have had time to collect and rise as completely as those in the interior parts.

In pneumatic-process steel, as commonly made, the elimination, such as it is, takes place in the ladle and molds, though when recarbonized in the vessel, a large part of the sonims are eliminated there.

When steel is "killed" so that it lies dead in the furnace, ladle, and molds, the conditions are in a sense unfavorable to the further riddance of sonims, because of the lack of agitation. Time will, of course, permit the floating to the top of any globules of sufficient size; but the formation of globules from the finely-divided sonims will proceed very slowly in dead steel. Stirring is as efficacious in agglomerating the fused sonims as it is in completing the reaction and collecting the precipitate in any other chemical operation. On such dead steel, made by what I have termed elsewhere the "solution" method, in which the escape of gas during freezing is nearly or quite prevented, no scum rises to the top of the ingot; and any sonims in the metal will remain there, though they will be unequally distributed throughout the ingot similarly to the segregated elements, as in the case of the Watertown ingots already noticed.

When the final additions to a charge of steel are made in the ladle, the conditions are unfavorable for the complete elimination of the sonims, though merchantable steel is made by following that plan. The time is limited in that case to the period which may elapse before the metal becomes too cold to cast successfully; while the average depth of metal through which a particle of sonim must rise in order that the metal may be

rid of it is several times greater than that of the charge in the furnace. If it be argued that, because of the great mass of a 50-ton heat, ample time may be allowed for the escape of the sonims, this greater depth of metal must be remembered, and the greater distance the entrained matter must travel to reach the slag. The greater depth is not of course proportional to the weight of charge; but it cuts down in a considerable degree the cleansing of the metal due to lapse of time after the final additions.

In the basic open-hearth furnace, the sonims may be eliminated by allowing time for the charge to work, and by having a slag low in iron; but greater care will be required than in the case of the acid open-hearth. There is one feature of the basic process, however, which aids their continuous elimination, namely, the reduction of metallic manganese from the slag by the carbon of the bath, the Mn entering the steel. This I never found to happen with acid steel. With the latter, once the Mn was all oxidized and eliminated from the metal, I never found it there until it was added in metallic form.

The practice of some steel-makers of adding a part of the Mn some time before tapping, and the remainder immediately before, or in the ladle, undoubtedly has an effect in eliminating the sonims by allowing more time for them to get into the slag after forming. Somewhat similar, in a lesser degree, is the practice of adding a part of the scrap to the molten bath. The resulting agitation, and the action of the Mn of the scrap, will cause early formation of fusible sonims, and therefore their removal.

Sonims are, and have been, commercially eliminated from steels in regular practice with tolerable results, even if the theory of the methods used has not been fully understood. "Working out," "shaking-down," and similar furnace-practices, as well as the final addition of Mn, are measures, the chief effect of which, aside from giving the steel the desired composition, is to allow or facilitate the elimination of sonims from the metal.

The various processes afford different degrees of completeness in the elimination of the sonims.

Well-made crucible steel should be the freest from sonims; electric steel next; then acid open-hearth steel finished in the

furnace; and finally open-hearth and Bessemer steel finished in the ladle.

In the first three processes named the practically complete removal of the sonims is possible, though it may be said not to be usual. In the absence of definite information as to what good slower driving would do, commercial reasons, or the demand for a maximum output at a minimum cost, usually outweigh the tendency to clean up the metal to the greatest practicable extent, and will continue to do so, at least, until proper specifications for sonims based on their proved effect are made and enforced.

In the crucible and electric steel processes there are or should be no final additions for the purpose of deoxidizing the metal; and therefore there will be formed, late in the history of the charge, no sonims which would have to be largely eliminated to secure the highest quality of product.

Among the remedies suggested for sonims, Howorth, in 1908, tried with some success the use of metallic sodium for the removal of O in iron and steel. Moissan suggested Mo as possibly useful for removing O from steel, because its oxide is volatile at steel-melting temperatures. Kent Smith claims for vanadium a certain "scavenging" effect on steel, including the elimination of oxygen and slag-like substances.

VI. Sonims in Alloy-Steels.

Aluminum.—The too-plentiful use of Al in steel may have been condemned, partly at least, because it forms oxides or silicates in the metal, which, being insoluble and infusible, exist in the solid steel as very harmful sonims. Of course, to form the oxide there must still be some oxide of iron or manganese in the steel. If the metal were free from O perhaps the weakening effect of Al when added in greater quantity than a few hundredths of 1 per cent. would not occur. So a part of the Al added in the ladle would form sonims which might be fluxed and floated out, while that added in the molds, if the steel were free from O, would not be oxidized, but would all be left to exercise its full effect in preventing the formation of gas-bubbles and the resulting blow-holes in the steel.

Without considering each alloying element separately, it may

be said that the possibility of their forming sonims should always be considered, particularly as to the following points:

1. What compound will they form if consumed, *i.e.*, oxides, silicates, sulphides, or others?

2. What are the physical properties of those compounds as to fusibility, specific gravity, and chemical relations at steel-melting temperature to the other substances present? Also, what would be their solvent power or solubility when molten with relation to the other sonims existing in the metal?

VII. Corrosion Due to Sonims.

About 25 or 30 years ago, the point was brought out that steel plates on ships corroded particularly through the electric action set up between pieces of adhering scale and the metal. The method may be illustrated by immersing a piece of newly-rolled plate-steel for some days in very dilute sulphuric acid. The metal around each piece of scale will be dissolved out much deeper than elsewhere, the pieces of scale remaining elevated above the surrounding surface like little table-lands. The metal is dissolved the faster, the nearer the place to the scale; and the latter is sometimes partly undermined, as it were, by the solution of the iron beneath its edges. As a result of this discovery the practice was adopted in the navy and elsewhere of removing all scale from steel ship-plates by pickling and brushing before they were worked into the ship.

While roll-scale is not a sonim, it has been shown that badly-made steels, presumably rich in sonims, are prone to corrode. Such steel is less homogeneous than well-made steel—a feature which Cushman gives as a condition favoring corrosion; and the sonims it contains represent what he calls "segregation," also a favoring factor.

Of the sonims it is natural to ascribe to iron oxide the first position as a cause of undue proneness to rust and corrode; but it is likely that some, if not all, of the others contribute thereto. It is probably the finely-divided sonims, rather than the larger globules, which help corrosion; and so it is evident that to make steel which will corrode slowly, it must be made so as to be good in other respects as well, particularly as shown by good physical properties, or, in other words, must be well

cleaned of its sonims of every kind. Corrosion may be expected to be more marked in steel made in the ladle than in that finished in the furnace or crucible.

VIII. Published Writings About Sonims.

Much interesting information has already been published regarding sonims, though limited for the most part to what could be learned by optical examination of finished steels and reasoning based thereon.

The earliest publication that I have found relating to sonims, or some of them, is the paper of Ferdinand Gautier, read in 1877 before the Iron and Steel Institute. He says that SiO_2 is produced, and afterwards an iron silicate, which remains interposed within the steel, and that the Mn allows the formation of a silicate of iron and manganese, which is much more fusible and passes into the slag. All of his theory regarding the matter might not pass to-day without criticism; but the substance of the quotation is still valuable, though too often ignored by steel-makers. Attention was apparently drawn at that time to the occurrence of silicates in steel by their known presence in wrought-iron. In the latter metal, however, their composition and manner of distribution are wholly different from those in any kind or grade of steel made by a fusion process. Consequently their effect is also different.

The following published papers relating to sonims in steel all contain valuable information.

1877. Ferdinand Gautier. Steel Castings. *Journal of the Iron and Steel Institute*, vol. xi., p. 40 (No. I., 1877).

1905. J. E. Stead. Sulphides and Silicates of Manganese in Steel. *Iron and Steel Magazine*, vol. ix., No. 2, p. 105 (Feb., 1905).

1905. Captain H. G. Howorth. The Presence of Greenish-Coloured Markings in the Fractured Surfaces of Test-Pieces. *Journal of the Iron and Steel Institute*, vol. lxviii., p. 301 (No. II., 1905).

1907. E. F. Law. The Non-Metallic Impurities in Steel. *Journal of the Iron and Steel Institute*, vol. lxxiv., p. 94 (No. II., 1907).

1909. Walter Rosenhain. "Slag Inclosures" in Steel. International Association for Testing Materials, Copenhagen Meeting.

1909. James E. Howard. Notes on Tests of Ingots and Derivative Shapes in Progress at Watertown Arsenal. *Proceedings of the American Society for Testing Materials*, vol. ix., p. 319 (1909).

1910. George Auchy. Mechanically Held Impurities in Steel. *Iron Age*, vol. lxxxv., No. 2, p. 108 (Jan. 13, 1910).

IX. Conclusions.

In view of the foregoing it seems evident that the entrained sonims are unique, having their own individuality, and are not to be confounded with anything else. To designate them as slag, mechanically mixed or otherwise, is misleading, since they are not slag, though their proper destination is the slag. It is true that iron oxide is often dissolved or absorbed from the slag to form sonims, but when in the metal bath, it is quite unlike the slag, except that it is non-metallic.

The mere fact alone that the analyses of sonims are different from those of slags would not prove that they were not slag, because entrained slag would have taken up SiO_2, Fe_3O_4, and MnO from the steel, cleansing it by so much, and being correspondingly changed in composition; but even then it could hardly have the 40 to 50 per cent. of Mn which the scums contain.

From the instances given of experiences from actual practice in which the sonims were both controlled and uncontrolled with profound results on the finished steel, it seems evident that the sonims exist in the metal in the steel-furnace and that the best practice demands their substantial elimination before the charge is teemed.

The assumption that has been made that sonims are due to oxygen absorbed and oxides formed while the heat is being cast is of doubtful accuracy.

A qualitative test in which they show sulphur is hardly good ground for the statement that sonims consist of manganese sulphide. Blast-furnace slag would show the same. The scum analyses do show, however, considerable S, which probably exists in the form of manganese sulphide.

The evidence thus points to the dissolved or very finely disseminated oxides and sulphides in the metal, together with those formed from the final additions, as the source of the entrained sonims in steel. The collection of these materials into globules begins (in the absence of metallic manganese) only after the final addition of manganese has been made, in part at least. The liquid sonims are sufficiently basic to take up some P from the metal, but whether in the form of a phosphide or a phosphate cannot now be told.

The excess of Mn which remains in Bessemer and open-hearth steel in metallic form is needed in common practice to secure quicker and more complete performance of its duty. Its effect on the physical properties of the finished steel is not particularly desirable, as the higher tensile strength it gives could be easily imparted by raising the carbon-content.

It is still a popular fallacy among some "tonnage" steel-makers that if steel contains an orthodox percentage of Mn no criticism of that feature of the composition will hold. Slight consideration is given by them as to whether the Mn exists in the steel as metallic manganese or in combination (which latter is indeed a minor point, as the Mn in the non-metallic form is so small a part of the whole), or to the greater question, whether or not the Mn has done its full work in eliminating the impurities, which is of supreme importance. It may be admitted on reflection that only the Mn which has been consumed, *i. e.*, combined with O or S, has been of use for purification; and it might also be claimed that, of the part consumed, only that which has escaped into the slag, or, in a less complete degree, that which has collected into somewhat large globules, has served its purpose. The usual chemical analysis makes no distinction between Mn in metallic form and that in the sonims.

Our knowledge of segregation in steel should be reviewed, so as to distinguish between the metallic part of the segregate and the non-metallic, or sonims.

Segregation is a feature of steels made by the oxidation processes, the Bessemer and open-hearth. It is of minor importance, if known at all, in steel made by the crucible process.

Many lines of study and experiment are seen which would add to our knowledge of sonims; and it may be maintained that there is no branch of steel-metallurgy which would yield to investigators more, or more useful, discoveries.

Method of Determining the Meridian from a Circumpolar Star at Any Hour.

BY EUGENE R. RICE, WICKENBURG, ARIZ.

(Canal Zone Meeting, November, 1910.)

THERE are many methods for determining the meridian, but all of those in common use involve at least two separate observations, one for latitude and one for azimuth. Such observations made upon a south star at any hour are open to the objection of possible inaccuracy due to ignorance of the absolute refraction, and also to slow change in the altitude of the star when near the meridian. The method of observing a circumpolar star at elongation, besides requiring a separate observation for latitude, is open to the serious objection that clouds may obscure the star at the critical moment. Moreover, a tedious wait for the star to come to elongation is sometimes necessary. Many of us can recall long, cold, weary vigils, waiting for Polaris to reach elongation.

For observing a circumpolar star at any hour, not only the latitude, but also the true time, must be known. This it is practically impossible to obtain, except in regions reached by the telegraph, and usually then one has to know the longitude also, in order to reduce "standard time" to "local time." The results obtained by this method, when the star is at culmination, are not very good, unless the true local time is absolutely known.

The methods of obtaining the meridian from the sun, by means of a solar attachment, or by direct observation, while good, are not absolute, by reason of refraction, change in declination of the sun, and small change in vertical angle, when the sun is near the meridian. For these two methods, latitude, time, and approximate longitude have to be known.

In the method here described, only one observation is needed, and from it are computed azimuth, latitude, and sidereal time. In other words, one can observe a circumpolar star—Polaris

preferred—at any hour of the night, and obtain exact results without knowing either latitude, longitude, or time. Briefly, the method consists of obtaining the true hour-angle of Polaris, its true altitude; and from this computing the azimuth of Polaris by means of formula 3.

With a helper to flash the cross-wires and call time—leaving the observer's hands free to manipulate the horizontal and vertical slow-motion screws simultaneously—an observation can be made in six minutes. The field-work consists essentially of simultaneously obtaining the true altitude of Polaris (the azimuth star), the true altitude of a fixed star with medium to small declination (the time-star), and the horizontal angle between Polaris and a mark. This is accomplished by taking double readings on the two stars at equal intervals of time, and taking the mean of the readings. For instance, in the example given (see form of note-book) the observer followed Pollux (the time-star) with the horizontal wire of the transit, while the assistant, with a watch, noted the time, and at exactly 11 h. 29 m. 00 s., the assistant called "Time."

The vertical circle was then read and the reading recorded.

The instrument—with plates set at zero—was then turned on Polaris (the azimuth star), and the observer followed Polaris with both the horizontal and vertical wires, until the assistant called "Time," at 11 h. 32 m. 00 s.

The vertical angle was then read and recorded, the upper motion unclamped, the transit turned on Station I., and the horizontal angle between Polaris and Station I. read and recorded.

The lower motion was then unclamped, the telescope plunged, the instrument again pointed at Polaris, and Polaris followed with the horizontal and vertical wires, by using the vertical and lower slow-motion screws, until the assistant called "Time," at 11 h. 35 m. 00 s.

The vertical angle was then read and recorded, the upper motion unclamped, and the instrument turned on Station I., the plate read and the reading recorded.

The instrument was then turned on Pollux, and at exactly 11 h. 38 m. 00 s., its vertical angle was obtained. Taking half the sum of the vertical angles of Pollux, half the sum of the vertical angles of Polaris, and half the last plate-reading, we

obtain the apparent altitude of Pollux, the apparent altitude of Polaris, and the horizontal angle between Polaris and Station I., at 11 h. 33 m. 30 s.

Referring to Fig. 1, it is evident, in the spherical triangle, *Z P S*, where *Z* is the zenith, *P* the pole, and *S* the time-star,

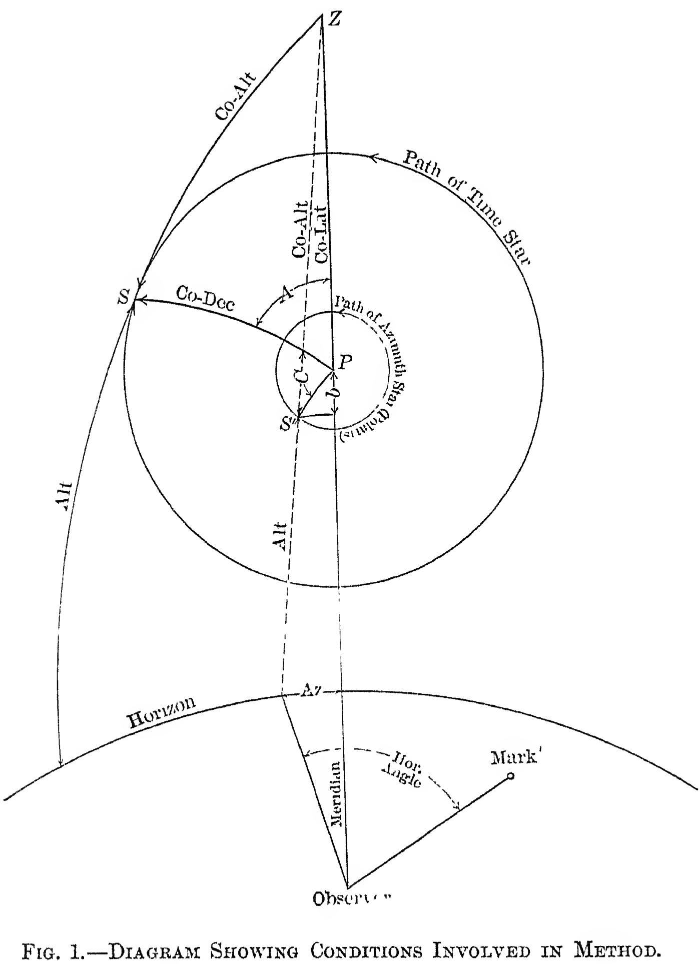


FIG. 1.—DIAGRAM SHOWING CONDITIONS INVOLVED IN METHOD.

that *A* is the hour-angle of the star, *Z P* the co-latitude, *Z S* the star's co-altitude, and *S P* the star's co-declination. It is evident that if the co-lat., co-alt., and co-dec. of the star *S* are known, its hour-angle *A* can be obtained by formula 1.

$$\cos \tfrac{1}{2} A = \sqrt{\frac{\sin S \times \sin (S - \text{co-alt.})}{\sin \text{co-dec.} \times \sin \text{co-lat.}}} \qquad 1$$

$$H = A + C. \qquad 2$$

$$\sin \text{az.} = \frac{\sin H \times \sin \text{co-dec.}}{\sin \text{co-alt.}} \qquad 3$$

$$\tan b = \tan \text{co-dec.} \times \cos H. \qquad 4$$

The altitude of the star is observed, its declination obtained from the Nautical Almanac or Table II., and the approximate latitude determined as follows:

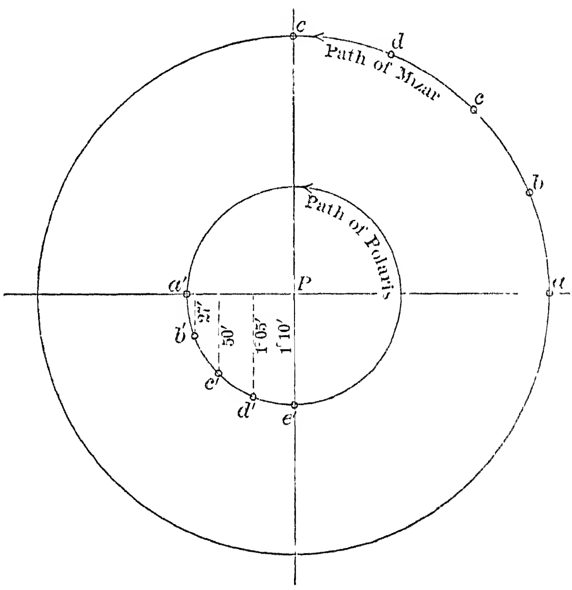


FIG. 2.—DIAGRAM SHOWING RELATIVE POSITION OF POLARIS AND MIZAR.

The latitude is equal to the elevation or altitude of the pole above the horizon, so it is evident that the observed altitude of Polaris is always within 1° 10′ of the latitude. Mizar (the next to the last star in the handle of the Big Dipper) and Polaris, are approximately diametrically opposite each other; when Mizar is at upper culmination, Polaris is at lower, and *vice versa*.

Mizar, in conjunction with Polaris, forms an excellent sidereal clock, and, by noticing the position of Mizar with respect

to Polaris, the approximate position of Polaris in its path around the pole can be determined.

Fig. 2 represents a view of the pole, showing Mizar and Polaris in their respective positions. It is not drawn to scale, on account of the small polar distance of Polaris. The illustration shows the relative positions as Mizar goes from eastern elongation to upper culmination, and Polaris from western elongation to lower culmination.

Thus, when Mizar has moved one-fourth of a quadrant above the pole, to *b*, Polaris has moved one-fourth of a quadrant below the pole, to *b'*, and then the altitude of Polaris, plus 27′, equals the approximate latitude.

Similarly, when Mizar has moved three-fourths of a quadrant, to *d*, and Polaris to *d'*, the altitude of Polaris, plus 1° 05′, equals the approximate latitude.

Thus, in the example given below, I estimated that Mizar had moved seven-eighths of a quadrant above the pole, hence the amount to be added to the altitude of Polaris to obtain the latitude was 1° 08′.

Table I. gives the latitude-correction for each quarter of a quadrant.

After determining A by means of formula 1, it remains to obtain the hour-angle, H, of Polaris.

The difference in hour-angle between Polaris and the time-star is practically a constant (C in Fig. 1), although it varies from year to year on account of the right ascensions of the stars, yet not varying by the same amount.

The right ascensions of the two stars are obtained from the Nautical Almanac, one subtracted from the other, and reduced to degrees, minutes, and seconds, which is C in Formula 2.

In the example given, the time-star is Pollux.

Right ascension of Pollux on April 16, 1910 =	7 h.	39 m.	48 s.
Right ascension of Polaris on April 16, 1910 =	1	25	39
Difference in right ascension =	6 h.	14 m.	09 s.

which equals 93° 32′ 15″ = C (see Tables II. and III. for April 16).

H is then substituted in formula 3, which is then solved for the azimuth.

A word of warning might not be amiss here. Formula 1 deals with the time-star only, and its co-altitude and co-declination are used in formula 1.

Formulæ 3 and 4 deal with Polaris: and its co-altitude and co-declination are to be used, and not those of the time-star. By choosing the right time-star and reading the position of Mizar fairly well, the resulting azimuth is far within the limits of error in reading an ordinary instrument.

If absolute accuracy is desired, after determining *H*, substitute it in formula 4, and add or subtract *b*, as the case may be, from the true altitude of Polaris, to obtain the true latitude; and use this new latitude in formula 1, and proceed as before.

This is unnecessary except in special cases, for with Polaris at lower culmination in latitude 34° N. and using Pollux as the time-star, if the latitude is in error 1° 10′ (the maximum possible error), the resulting error in azimuth is less than 40″;—this, too, when Polaris is changing its azimuth most rapidly. By choosing the proper time-star and making approximate latitude-correction, the resulting error in azimuth may be reduced to very few seconds;—far less than can be read by doubling with a transit graduated to 30″.

The time-star nearest elongation should be chosen. In testing this method, Polaris was observed near culmination, because it is then the maximum error would come in, and it could thus be shown how the method would work under the most adverse conditions.

Example from Note-Book.

Object.	Time.			Vertical Angle.			Plate.			Instrument.	Remarks.
	Hr.	Min.	Sec.	Deg	Min.	Sec.	Deg.	Min.	Sec.		
Pollux.....	11	29	00	29	11	30	....	....	....	Erect.	
Polaris....	11	32	00	32	55				00	Erect.	Mizar at
Station I..	....	....	....	....	....	....	84	51	30	Erect.	+ $\frac{3}{8}$
Polaris....	11	35	00	32	54	30	84	51	30	Inverted.	April 16,
Station I..	....	....	....	....	....	....	169	42	15	Inverted.	1910.
Pollux.....	11	38	00	27	24		....	....	....	Inverted.	

Average apparent altitude of Pollux, . .	28°	17′	45″
Less refraction, ...		1	42
True altitude of Pollux, . . .	28°	16′	03″

Average apparent altitude of Polaris, . .	32°	54′	45″
Less refraction, ...		1	29
True altitude of Polaris, . . .	32°	53′	16″

For Mizar at $+\frac{7}{8}$, the amount to be added to altitude of Polaris (see Table I.) for latitude, is 1° 08′.

Therefore we have for formula 1 : Log sin S = 9.9999953

Lat. = 34° 01′ 16″	Co-lat. = 55° 58′ 44″	Log sin (S — co-alt.) =	9.6716093
			9.6716046
Alt. = 28 16 03	Co-alt. = 61 43 57		9.8634043
			9.8082003
Dec. = 28 14 46	Co-dec. = 61 45 14	Log cos $\frac{1}{2}$ A =	9.9041001

$2S$ = 178° 86′ 115″		$\frac{1}{2}A$ = 36° 41′ 30″
S = 89 43 57		A = 73 23 00
Co-alt. = 61 43 57		
S — co-alt. = 28° 00′ 00″		

Log sin co-lat. = 9.9184663
Log sin co-dec. = 9.9449380
9.8634043

$H = A + C$
C = 93° 32′ 15″ (See Table III.)
A = 73 23 00
H = 166° 55′ 15″

$$\text{Sin az.} = \frac{\sin H \times \sin \text{co-dec.}}{\sin \text{co-alt.}}$$

H = 166° 55′ 15″		Log sin H =	9.3546791
Dec. = 88° 49′ 33″	Co-dec. = 1° 10′ 27″	Log sin co-dec. =	8.3115767
			7.6662558
Alt. = 32 53 16	Co-alt. = 57 06 44	Log sin co-alt. =	9.9241427
		Log sin az. =	7.7421131
		Az. = 18′ 59″	

Horizontal angle between Polaris and Station I. $= \dfrac{169° \ 42′ \ 15″}{2}$

= 84° 51′ 07″
Less 18 59

N. 84° 32′ 08″ E. = Bearing between instrument and Station I.

To get the true latitude, substitute in formula 4, and solve for b.

Log tan co-dec.	=	8.311668
Log cos H.	=	9.988586
Log tan b.	=	8.300254

b.	=	1°	08′	38″
Alt.	=	32	53	16
True lat.	=	34°	01′	54″

By using this true latitude, we can re-solve; but the azimuth comes out 18′ 59″, as above obtained.

The bearing of the line to Station I. was determined a year or so ago by separately observing Polaris and 51 Cephi at elongation, and repeating the angle four times on a transit reading to 1′. By splitting, the final reading could be got to 30″. The bearing of this line is correct to within 10″.

The bearing by Polaris was	N. 84° 32′ 06″ E.
The bearing by 51 Cephi was	N. 84 32 12 E.
Average,	N. 84° 32′ 09″ E.

Using a new transit, reading direct to 30″ and able to split to 15″, I obtained by the method described in this paper the following results:

		Time-Star.
Polaris near lower culmination, . .	N. 84° 32′ 10″ E.	Pollux.
Polaris near lower culmination, . .	N. 84 32 08 E.	Pollux.
Polaris near lower culmination, . .	N. 84 32 01 E.	Pollux.
Polaris half way between culmination and elongation, ...	N. 84 32 13 E.	η Ursae Majoris.
Average, . . .	N. 84° 32′ 08″ E.	

In all these the latitude was determined by the approximate method.

Table I. gives the latitude-correction to be applied by estimating the position of Mizar.

Table II. gives the declination and right ascensions of Polaris, Pollux, α Ursae Majoris, and η Ursae Majoris for the 1st, 10th, and 20th of each month of 1910, and mean annual change of same.

Table III. gives the values of C for each of the above stars, and the mean annual change.

The data in Tables I. and II. will be good for many years to come. The calculations may be shortened by using values to the nearest minute in formula 1.

The method as here described probably appears laborious, but I have attempted to explain it in detail, so no one would have trouble in using it. After practice, it is exceedingly simple. To one accustomed to the use of logarithms, the reductions give no trouble, as there is only one cosine in the two formulas, all the rest being sines. In choosing the time-star, the one nearest elongation should be taken. To those unfamiliar with the stars named, Fig. 3 will be of service.

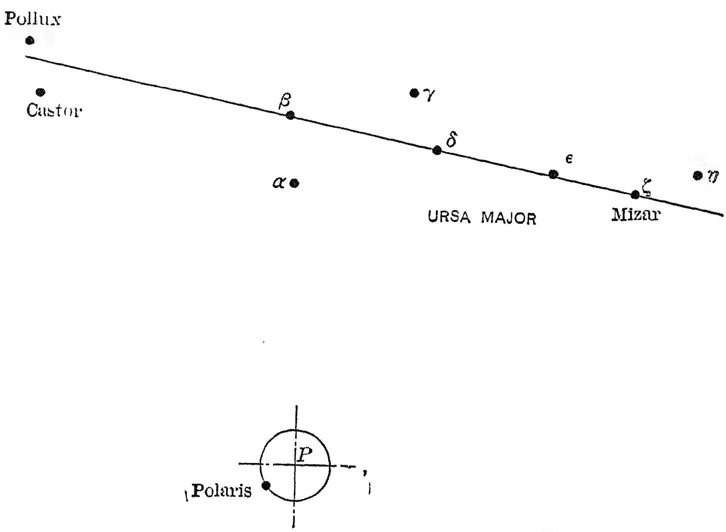


FIG. 3.—DIAGRAM SHOWING RELATIVE POSITIONS OF STARS MENTIONED.

TABLE I.—*Latitude-Correction for Estimated Relative Position of Mizar.*

Position of Mizar.	Correction.
Even with Pole	0
¼ above or below	± 27′
½ above or below	± 50′
¾ above or below	± 1° 05′
Directly above or below	± 1° 10′

Table II.—*Right Ascension and Declination of Certain Circumpolar Stars for the Year* 1910.

Date		Polaris.		Pollux.		α Ursae Majoris.		η Ursae Majoris.	
		Right Ascension.	Declination.	Right Ascension.	Declination.	Right Ascension.	Declination.	Right Ascension.	Declination.
		Hr Min Sec.	Deg Min Sec	Hr Min Sec.	Deg Min Sec.	Hr Min Sec	Deg Min. Sec.	Hr Min Sec.	Deg Min Sec
Jan.	1	1 26 52	88 49 49	7 39 49	28 14 41	10 58 11	62 13 63	13 43 59	49 45 32
	10	43	50	49	41	12	63	59	30
	20	31	50	49	42	12	64	60	28
Feb.	1	18	50	49	42	13	65	60	27
	10	09	49	49	43	13	67	60	27
	20	00	48	49	43	14	69	61	27
March.	1	1 25 53	46	49	44	14	72	61	28
	10	47	44	49	45	14	74	61	30
	20	42	41	49	45	14	77	62	32
April.	1	39	37	49	46	14	80	62	34
	10	38	35	49	46	14	82	62	36
	20	39	32	48	47	13	84	62	39
May.	1	43	28	48	47	13	86	62	42
	10	47	26	48	47	13	88	62	44
	20	54	24	48	47	12	89	62	47
June.	1	1 26 03	22	48	47	12	89	62	49
	10	11	21	48	46	12	89	62	51
	20	21	20	48	46	12	89	61	52
July.	1	33	20	48	46	11	88	61	53
	10	42	20	48	45	11	86	61	54
	20	53	21	48	45	11	84	61	54
Aug.	1	1 27 05	22	48	44	11	82	61	54
	10	13	24	49	44	11	79	60	53
	20	23	27	49	43	11	77	60	52
Sept.	1	33	30	49	42	11	73	60	50
	10	39	33	49	41	11	72	60	48
	20	44	36	50	40	11	67	60	45
Oct.	1	49	41	50	40	11	63	59	42
	10	52	44	50	39	12	60	59	39
	20	53	48	51	38	12	57	59	36
Nov.	1	52	53	51	37	12	53	59	32
	10	50	56	51	36	13	50	59	28
	20	47	59	52	35	14	48	59	24
Dec.	1	40	88 50 03	52	35	14	46	59	21
	10	33	06	52	34	14	45	60	17
	20	25	08	53	34	15	44	61	14
Annual Change		+27.36 Sec.	+18.62 Sec.	+3.68 Sec.	−8.5 Sec.	+3.73 Sec.	−19.39 Sec.	+2.37 Sec.	−18.04 Sec.

TABLE III.—*Values of C for the Stars of Table II. for the Year* 1910.

Date.		Pollux.			α Ursae Majoris.			η Ursae Majoris.			Date.		Pollux.			α Ursae Majoris.			η Ursae Majoris.		
		Deg	Min	Sec	Deg.	Min	Sec	Deg	Min	Sec			Deg	Min	Sec.	Deg	Min	Sec	Deg	Min	Sec.
Jan.	1	93	14	15	142	49	45	184	16	45	July	1	93	18	45	142	54	30	184	22	00
	10	93	16	30	142	52	15	184	19	00		10	93	16	30	142	52	15	184	19	45
	20	93	19	30	142	55	15	184	22	15		20	93	13	45	142	49	30	184	17	00
Feb.	1	93	22	45	142	58	45	184	25	30	Aug	1	93	10	45	142	46	30	184	14	00
	10	93	25	00	143	01	00	184	27	45		10	93	09	00	142	44	30	184	11	45
	20	93	27	15	143	03	30	184	30	15		20	93	06	30	142	42	00	184	09	15
March	1	93	29	00	143	05	15	184	32	00	Sept.	1	93	04	00	142	39	30	184	06	45
	10	93	30	30	143	06	45	184	33	30		10	93	02	30	142	38	00	184	05	15
	20	93	31	45	143	08	00	184	35	00		20	93	01	30	142	36	45	184	04	00
April.	1	93	32	30	143	08	45	184	35	45	Oct.	1	93	00	15	142	35	30	184	02	30
	10	93	32	45	143	09	00	184	36	00		10	92	59	30	142	35	00	184	01	45
	20	93	32	15	143	08	30	184	35	45		20	92	59	30	142	34	45	184	01	30
May.	1	93	31	15	143	07	30	184	34	45	Nov.	1	92	59	45	142	35	00	184	01	45
	10	93	30	15	143	06	30	184	33	45		10	93	00	15	142	35	45	184	02	15
	20	93	28	30	143	04	30	184	32	00		20	93	01	15	142	36	45	184	03	00
June.	1	93	26	15	143	02	15	184	29	45	Dec.	1	93	03	00	142	38	30	184	04	45
	10	93	23	30	143	00	15	184	27	45		10	93	04	45	142	40	15	184	06	45
	20	93	21	45	142	57	45	184	25	00		20	93	07	00	142	42	30	184	09	00
Annual Change		— (5′ 55.3″)			— (5′ 54.5″)			— (6′ 15.0″)			Annual Change		— (5′ 55.3″)			— (5′ 54.5″)			— (6′ 15.0″)		

The Conference Department at Lehigh University.

BY DR. HENRY S. DRINKER,* SOUTH BETHLEHEM, PA.

(Canal Zone Meeting, November, 1910.)

FEW men reach middle life without having had the experience of failure in one or more undertakings; and most of us can look back with gratitude to help or advice given us by friends at critical periods of our lives. If, as men, we ask and take the benefit of such aid, why should we expect our sons to be stronger? A parent can make no greater mistake than to send a boy, however steady and intelligent he may be, off to school or college, and expect the boy always to cope success-

* President of Lehigh University.

fully with all the new conditions and new difficulties that may confront him. This is undoubtedly more the case with a freshman, in his first year at college, than in later years, or than it is with a boy at day- or boarding-school, where the boy receives a greater degree of care and oversight than can be given to the young man entering college; of all periods in life, the youth in his freshman year—fresh from the overshadowing care of home and school—may need judicious advice and timely aid. The change from the restraint of home and school to the freedom of college life is sudden and marked, and it is in this the first year of the boy or young man of 17, 18, or 19 years of age, that some tactful, judicious aid should be near him in his work and course. Some boys go to college so well prepared that they become careless and finally fall behind in their work from over-confidence. Others meet trouble from ambitiously trying to do more than their previous preparation, or physical or mental strength, can sustain. Others fall behind from illness—from injuries—from natural inability to grasp at first, without more aid than comes to them in the regular course of instruction, the higher mathematics—or mayhap the modern languages, or other subjects presented by the advanced curriculum of the present day. When a boy or youth begins to fall off in his work is the time to apply the remedy, not later, when the harm may be irremediable; and the parent at home, or the head-master at boarding-school, or the freshman advisory committee of the faculty at college, should be ready with "First Aid to the Injured."

The easiest way for the faculty to deal with deficient scholarship is undoubtedly to set some standard, say the passing periodically of a fixed percentage of the work required, without which the student is dropped; and this rule, of course, reduces the practice to a simple clerical matter of figures—the student dropping out of the subject, or out of college, if he does not reach the standard. Yet this is but applying the test of the old Procrustean bed to modern higher education; and who will venture to deny that often, out of two men who fail in a test, one may really be incapable of meeting the work, while the other may be physically and mentally so strong that with a little aid and care, judiciously given at the right time, he may be able to overcome his difficulties and save the precious year

or years in life that dropping back might entail. It should be a matter of individual, careful, interested scrutinizing in each case. This may be impossible in an institution numbering thousands of students, though at Princeton the tutorial system has been instituted to meet it; but in colleges carrying, by preference, a limited student body (the minor colleges, as they are called), it is entirely practicable.

At Lehigh, where the student roll is restricted and held to an average of not exceeding 200 in a class, to the end that individual attention may continually be given to each man, the question of how best to deal with this matter—how to keep a man in college after he has been once deemed fit to enter, instead of throwing him out as soon as he shall fall below a certain standard, for any cause—has been a matter of careful study, not only by the faculty but markedly by the alumni body. At all colleges a certain amount of coaching is sought by students requiring help. As a rule, faculties object to this coaching being done by instructors teaching in the departments, who undertake such extra work for pay. It is apt to lead to abuses, or to the suspicion, at least, that the student's path is made thorny to the end that he shall have to pay for aid to clear the way; and it is very doubtful whether the regular teacher should be encouraged or allowed to put into this outside work the energy which should be conserved and concentrated on his proper work. It is best that as a rule the teaching force shall not be called on to do private teaching in the nature of coaching. But this results in the establishment of coaching-schools, which at some places have reached such a point of development that they are said to be highly-paying institutions, the fees running from $1 to $5 per hour, and the work done being systematically an attempt to cram and shove the student through an impending quiz or examination by the help of another man's brains, rather than the giving of judicious aid to enable the student to use best his own brains and to develop properly his study-power. Moreover, it becomes a resource of the rich rather than of the poor student. In short, it is generally all wrong in theory, practice, and the results attained.

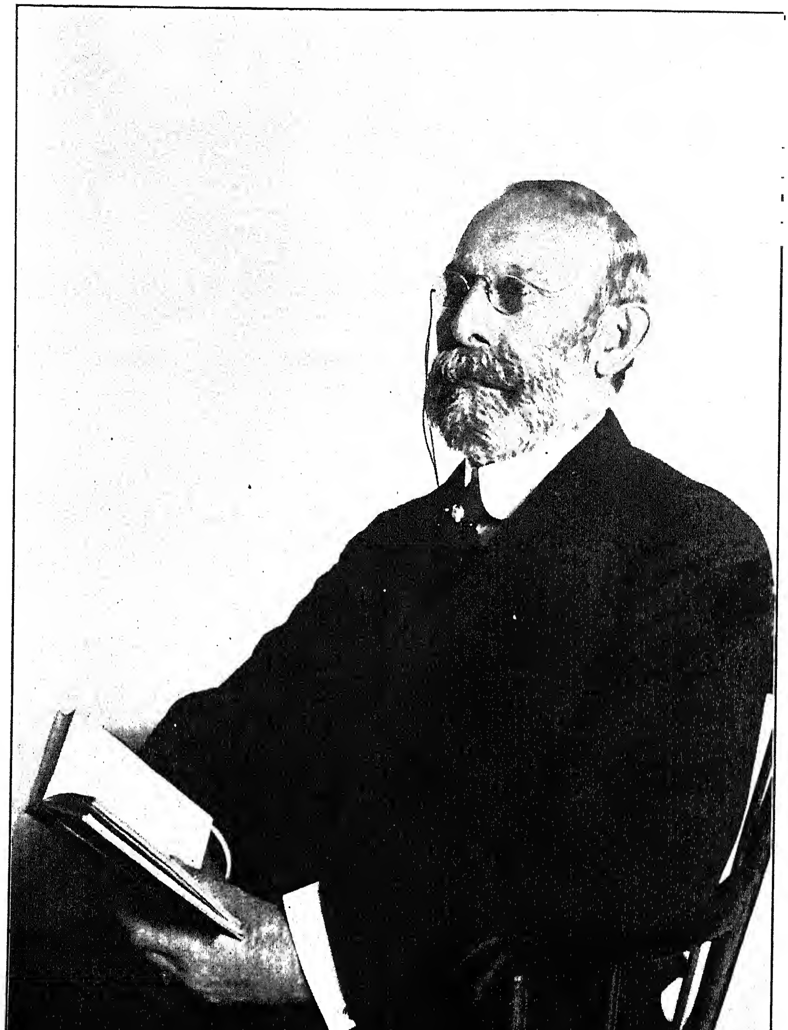
But while this remedy may be the wrong one, the weakness is there, and some remedy is needed. At Lehigh, we believe that for our younger students we have met this need by the

institution, two years ago, of our conference school, organized with an experienced professor of mathematics at its head as director, a man heartily in sympathy with the alumni movement in this matter—the chairman of the freshman advisory committee, and not only an exceedingly good teacher, but a very human man with a large fund of sympathy and love for young men. With him are associated experienced teachers in modern languages, chemistry, and physics, these four subjects being those of our curriculum in which experience has shown that young students are most likely to require aid. As teachers in these departments they know how each department wishes its work done; but their labors in the college are so apportioned that the work in the conference school shall not overburden them, and they are paid by the university for this conference work—not by the students—so that they have no interest or concern in the number of men applying for aid or in the fees paid.

This department is open to all sophomores and freshmen desiring aid or advice, from 7 to 9 p.m. on Monday, Tuesday, Thursday, and Friday nights, for a fee of $1.50 for the four lessons, and it sits from 2 to 4 p.m. on Wednesday and Saturday to give instruction during these hours free of any charge to students applying.

In this department the young student studies under the care of an experienced teacher, who is ready to elucidate with blackboard and pencil any difficult subject; and the student instead of being crammed, is made to study and think and reason for himself, with such help only as he should essentially have to master the subject. It is a great mistake, in many cases, to blame a boy for falling behind in his work. We all fall back sometimes. We all at times need help and encouragement; and we should not exact from our boys a greater degree of infallibility in work than we as men measure up to, nor should the boy be dropped until after the scrutiny and care of a competent adviser has demonstrated that either the boy's incapacity for study, or his carelessness in work, is such that he is really unfit to go on, is, in fact, not college timber. As soon as that fact has been demonstrated he should of course be promptly dropped—for his own good, and for the good of his college mates who may be unfavorably affected by his influence and example.

To face p. 837.



Chas. B. Dudley

Biographical Notice of Charles B. Dudley, Ph.D.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910)

In the long list of our illustrious and lamented dead, there are names which recall personality as well as career and achievement; social as well as scientific merit; devotion to the Institute itself as well as to some one of the professions which it represents. These men we remember because they did not forget us; besides our pride in their wider work and fame, we are proud of what they did with and through us; especially do we rejoice if our early recognition of them opened the way to a recognition by the world; and in their departure we claim to sorrow, not merely as members of an appreciative public, but also as bereaved personal friends. In common with thousands, we regret our loss, but we do not regret only—we mourn and miss. To this group of those whose death has brought to us the double sense of public and private grief belonged our strong, bright, dear brother, in whose memory this inadequate tribute is offered.

Charles Benjamin Dudley was born July 14, 1842, at Oxford, Chenango county, N. Y. He came on both sides of New England blood, his father being from New Hampshire and his mother from Connecticut. Emigrating from the rocky hills of their native States, they settled upon the more fertile soil of what was then regarded as the West, and, with thousands of other pilgrim sons, imitating the pilgrim fathers, contributed to the foundation of new commonwealths. The Empire State owes much to this element of its population.

Dudley's early education was obtained, like that of the sons of the pioneers generally, by attending the village school and academy in the autumn and winter, while he worked upon the farm during the rest of the year. But his ambition looked forward to a college-course; and this was almost within his reach when a passion still stronger took possession of him. The out-

break of the War of the Rebellion appealed to his patriotic enthusiasm; and, in 1862, at the age of 20, he enlisted as a private in the 114th New York Volunteers. During the following three years he took part in seven battles, gaining his experience of the routine as well as the adventure of war in all parts of the country, but mainly in the Southwest, and under circumstances of discomfort and discouragement most trying to the patience and courage of our soldiers. He was at the siege of Port Hudson in 1863, and went through the disastrous Red River campaign of the spring of 1864. But, after that campaign, he was transferred to Virginia, where, on Sept. 19, 1864, at the battle of Opequon Creek, fought near Winchester, and not long before the more famous battle to which Winchester gave its name, he received a severe bullet-wound, which rendered him incapable of further active duty. It left him, indeed, partially crippled for life, with a bullet in his leg as a perpetual reminder of the source of his disability.

But Dudley was one of those citizen soldiers who went into the War, not because they loved war, or expected to make it a life-vocation. They undertook military service as a patriotic task, yet even when, in the performance of it, they became veteran soldiers, they did not become professional soldiers. Their hearts were in their homes; and their plans for the victories of peace were only temporarily postponed, not superseded, by the experiences of the camp. This is the explanation of the phenomenon, so often named with wonder, of the swift re-absorption of the Grand Army, at the close of the War, into the ranks of the people from which it had sprung. The soldiers went home, where they had always longed to be, and resumed the work which they had laid aside at the call of a higher duty. At least, this is true of Dudley, and, as I believe, of many like him. As an illustration of his attitude towards the work of a private soldier, which he performed with as much fidelity and gallantry as if it had been his life-work, I may recall that, in 1863–4, when the Union forces in Louisiana went into winter-quarters, he had his Latin grammar and reader sent to him from home, and spent in the study of Latin the spare hours of that period. Unfortunately, through a capture of the stored camp-baggage, after active operations had been renewed, these books became the property of the enemy;

but the enemy had little opportunity, after that, to make use of the classics, and Dudley, meanwhile, had put the contents of the books where they could not be captured, unless he were captured with them.

Upon his discharge in 1865, he returned at once to his rural home, and took up his life again, just where he had laid it aside. I had almost written, "laid it *down*"—and if I had done so, I could have defied the critic and the proof-reader, for he had as truly laid it down as if the sacrifice had been accepted and ratified by death. That the life thus freely offered was returned to him, did not impair the completeness of the sacrifice. His escape from the death he had dared was only a Voice which said, "I have further use for thee!" So, at least, I understand it: and so, at least, did he! And I must add here, as a part of my picture of the man, that he never "traveled" on his military record. He did not run for office on the strength of his army service, or urge it as a claim to civil distinction or preference. I think he regarded it as an episode, such as every good citizen should meet like a man—without bragging about it afterwards. I think he would as soon have paraded his honesty as his patriotism. Such things, in his view, "go without saying."

So the crippled veteran of 23 went back to the Oxford Academy to prepare for college, with nothing to show in that line for the three years he had lost except what he had managed, in camp, to get out of his Latin grammar and reader, and, possibly, also a certain amount of fidelity, patience, and indomitable perseverance and courage, which had been both revealed and trained in the service of his country. And in 1867, as a man of 25 years, he modestly entered the Freshman class at Yale, along with a host of youngsters, who had not wasted any time in soldiering. Moreover, he did not cut short his college course on the ground that so much time had been lost, but went through the whole of it, shirking nothing, just as if it had been simply another campaign, and finally, in 1871, took his degree as A.B. with his class.

I make no apology for dwelling at such length upon these mere preliminaries of Dr. Dudley's career. For, to my mind, the secret of the man's character, and of his consequent success, is indicated in these very things.

I have known wise parents many, who, being able to give their sons all the advantages of a thorough education, have put them through a complete collegiate course as a preliminary to a professional course, before launching them into active and independent life. Happy the father, whose son is willing to wait so long, and is not spoiled meanwhile by scholastic life for the life of work among men which is to follow! And happy the son, who understands the value of such a preparation, and whose father is able and willing to pay the bills! (According to my observation, the wise fathers are more numerous than the wise sons!)

But I do not recall a single instance, except that of Charles B. Dudley, in which a youth has framed for himself a plan of thorough education of that sort, has "lost" three precious years in obedience to a higher duty, and has then returned to the prosecution of his original plan, without counting at all the time thus "lost."

To complete the picture of this extraordinary case, I should say that, when Mr. Dudley was graduated at Yale in 1871, he had no money to pay for the further schooling which he desired, and was, besides, in debt for money which he had been obliged to borrow to enable him to finish his college-course, although the amount thus owed was relatively small, because he had paid a large part of his expenses by outside work of all kinds. Under these circumstances, he did not look about for some patron who would lend more money in aid of a promising student, but set himself at once to earn what was required, both to repay his debt, and to furnish him for future progress. In other words, having already given three years to his country, he now gave another year to his personal honor. This year he spent in work as night-editor of the New Haven *Palladium* and the *Yale Courant*, successively, and in other employments, which enabled him to achieve at last his cherished ambition, and enter the Sheffield Scientific School of Yale University, from which he was graduated in 1874 as Ph.D. He had devoted himself, during his two years' course, to study and practice in chemistry; and his graduating thesis, afterwards published in full abstract in the *Proceedings of the American Association for the Advancement of Science*, was on "Lithium, and a Glass Made with Lithium."

At this point, I would pause again to call attention to the exceptional character of this example. Here is a man who has become 32 years old before entering upon his professional work. The delay has been due to intervals of three years and one year, sacrificed by the man himself, at the call of what he deemed higher duty—the first to patriotism, the second to personal honor. But the original plan of the youth has been carried out, irrespective of these interruptions, without one iota of abridgement or change; and the man stands at 32 where he might have stood at 28, or even (if he had decided to skip his classical college-course, and plunge at once into his technical course) at 24.

Yet, does he really stand only where he would have stood four or eight years earlier, if no time had been "lost"? Is it nothing that a man, having been a soldier, knows how to be a comrade—to meet and understand and win to hearty co-operation other men? Is it nothing that, having gained through a classical education the ability to comprehend the lessons of history and the inherited associations which actuate, even unconsciously, the men of to-day, he is able to communicate convincingly to others what he knows himself? Is it nothing that he has shown himself to be a lover of his vocation, yet capable of sacrificing even his vocation to duty and honor? Finally, is his certified professional ability less likely to win professional employment and success, when backed by that other certificate, furnished by a wide experience with men and their enterprises? These and other like questions are answered by the subsequent career of Dr. Dudley, which illustrates the deep utterance of Browning: "Get thy tools ready; God will find thee work."

The first year after his graduation from the Sheffield School he spent as Assistant to Dr. George F. Barker, Professor of Physics at the University of Pennsylvania; and his success in this position led to a call, which he accepted in September, 1875, to the position of teacher of science in the Riverside Military Academy, of Poughkeepsie, N. Y. But before entering upon the duties of this position, he resigned it to accept the place thenceforth the center of his activity and influence—namely, that of Chief Chemist of the Pennsylvania Railroad Co., which he assumed Nov. 10, 1875, and held until his death.

During his service as Professor Barker's assistant, he had given, so far as I am aware, no evidence of his general interest in the progress of applied science, except the publication in the *Franklin Institute Journal* of some translations of German technical papers; and what directed to him the attention of the officials of the Pennsylvania Railroad Co., I do not know. Very likely some one of the keen, wise managers of that company had met and measured the man himself. At all events, this is certain: they made a fortunate choice when they chose him.

At that time the employment of chemists, or other professional experts, by great corporations was, though not altogether unknown as a temporary expedient, scarcely recognized as a part of the regular organization of such enterprises. Whatever partial precedents may be adduced, it cannot fairly be denied that the appointment of Dr. Dudley by the Pennsylvania Railroad Co., coupled, as it was, with the provision of ample laboratory-facilities for him, and, above all, with a sincere recognition of the value of his advice and an uncompromising execution of his recommendations in the execution of business contracts, was an epoch-making event. And the choice of the right man for the place was fortunate, not only for the company and for him, but also for many other companies, whose course was influenced by the result of this experiment, and for many other experts who were afterwards, in consequence of this successful example, similarly employed as the permanent scientific advisers of great corporations.

When the Pennsylvania Railroad Co. created the experimental and advisory department at the head of which Dr. Dudley was placed, it was purchasing enormous quantities of materials, ranging from iron and steel to oil, paint, and soap, but it had no scientific and systematic means of testing these materials before using them, or of framing specifications in advance for the guidance of manufacturers. Indeed, the maintenance of a separate department for criticism, investigation, and suggestion, in connection with any large enterprise, was practically new. The leading railroad companies had set the example by their thorough auditing departments; and the establishment of such a department, of even wider scope, by Carnegie, Phipps & Co., had proved that a great manufactur-

ing concern could profitably employ, in merely critical and advisory work, an expensive crop of "non-productive" skilled laborers, keeping accounts not merely with individuals but with furnaces, processes, and raw materials. But the Pennsylvania Railroad Co. was itself a manufacturer, as well as a consumer; and Theodore N. Ely, the Superintendent of its motive power, conceived the plan of an auxiliary department of tests and experiments, including a chemical laboratory. Dr. Dudley's work as the Chief Chemist and head of that department occupied the remaining thirty-five years of his life, and made him useful and famous throughout the world. He discovered and perfected tests, or prepared specifications, for all important materials used by the company, such as coal, water, lubricating-oil, paint, steel, iron, etc. It is reported that there are now between forty and fifty sets of specifications, physical or chemical or both, for the chemical provisions of which Dr. Dudley was directly responsible, while his advice largely influenced the physical provisions. The chemical laboratory employs twenty-seven persons, of whom twenty-three are regularly educated chemists, one being also a trained bacteriologist, who is occupied in the examination of water-supplies, disinfectants and their use, and tests for medical diagnosis required in the administration of the company's Relief Fund. The number of tests and chemical determinations made in connection with materials offered for acceptance amounts to about 60,000 per annum. The story of the ventilation of passenger-cars, investigated in this laboratory, and resulting in the development of a satisfactory system, furnishes one of the many instances in which the benefits of this institution have extended far beyond the business of the corporation which it primarily serves. Another instance is the exhaustive study of explosives, which bore fruit in the formulation of rules governing the transportation of these and other hazardous materials.

In the course of his experiments and investigations, Dr. Dudley made many inventions and discoveries, for the great majority of which he never took out patents, "having," as he used to say, "no time for that." One of them, the design of the wick and burner for a railroad-lantern, is now in use everywhere. In another class of his discoveries—namely, the perfection of rapid and accurate commercial tests and analyses—

belongs the origination of a method for the detection of the presence of cotton-seed oil as an injurious adulterant of liquid lubricants.

In the conduct of his great work, Dr. Dudley pursued the policy of a frank contribution and exchange of the knowledge obtained. I do not mean to say that he kept nothing secret, in the business interests of his employers, or in fairness to others. But, instead of adopting the easy course of saying nothing if he could not tell everything, he obeyed a wise as well as generous impulse to make known as much as he could. The result was not only most creditable to him, but most helpful to his special work. No better example can be given than that of his activity as a member of this Institute, which he joined in 1878, perceiving the possibilities of world-wide professional co-operation through such a medium.

He lost no time in proving his desire for such fraternal exchange; for he was not one of those who hoped to get without giving—like a man who wrote me, not long ago, that, after having been a member for a number of years (during which he had made no contributions to the *Transactions*), and having found nothing in our publications bearing upon his specialty, he thought he might as well resign! To say nothing of the selfishness of such a view, it is clearly short-sighted. He who would hear something ought to say something. Mere air may rush to fill a vacuum; but professional co-operation does not. Let Dr. Dudley's history point this moral!

In 1878, as a new member of the Institute, without waiting to get light by simply "keeping dark," he contributed to the *Transactions* his paper on The Chemical Composition and Physical Properties of Steel Rails,[1] embodying the results of numerous experiments and much thought. That contribution "fired a shot heard round the world." The pages of our *Transactions* bear abundant witness to the extent of the interest which it aroused and the value of the discussion which it elicited. The theories and formulas which it tentatively proposed have been modified or superseded since, partly by his own subsequent investigations; but it placed its author in the front rank of investigators, and gave to the Institute a leader-

[1] *Trans.*, vii., 172 (1878–79).

ship in that field not yet wholly lost. Whoever studies the relation of the physical and chemical factors in steel, and the history of its scientific elucidation, will find himself repeatedly referred to Dudley's great paper and its discussion; and no better summary of the latest results of the long debate can be found than his address, Some Features of the Present Steel Rail Question, delivered before the American Society for Testing Materials in 1908.

The following list of Dr. Dudley's contributions to the *Transactions* indicates their range and value. His ability and zeal were promptly recognized by his fellow-members, who elected him in 1880 a Vice-President of the Institute.

Contributions.

Papers.

The Chemical Composition and Physical Properties of Steel Rails. Vol. VII., p. 172.

Does the Wearing Power of Steel Rails Increase with the Hardness of the Steel? Vol. VII., p. 202.

Wearing Capacity of Steel Rails in Relation to Their Chemical Composition and Physical Properties. Vol. IX., p. 321.

(C. B. Dudley and F. N. Pease) Notes on the Constitution of Cast-Iron. Vol. XIV., p. 795.

The Wear of Metal as Influenced by Its Chemical and Physical Properties. Vol. XIX., p. 892.

The Making of Specifications for Structural Materials. Vol. XXI., p. 379.

Standard Specifications for Cast-Iron Car-Wheels. Vol. XXXV., p. 189.

Discussions and Remarks.

Of his Rail papers in Vol. VII. Vol. VII., pp. 393, 413.

Of his Rail paper in Vol. IX. Vol. IX., p. 588.

Of Witherow's paper on The Clapp-Griffiths Converter. Vol. XIV., p. 938.

Of Langley's paper, International Standards for the Analysis of Iron and Steel. Vol. XIX., p. 635.

On Magnetic Concentration of Iron-Ore. Vol. XX., p. 580.

Of Hunt's paper, Tests and Requirements of Structural Wrought-Iron and Steel. Vol. XX., p. 703.

Of Campbell's paper, The Influence of Carbon, Phosphorus, Manganese, and Sulphur on the Tensile Strength of Open-Hearth Steel. Vol. XXXV., p. 1046.

Of Gayley's and Johnson's papers on Blast-Furnace Practice. Vol. XXXVI., p. 792.

Of Roe's paper on Manufacture and Characteristics of Wrought-Iron. Vol. XXXVI., p. 811.

The foregoing list represents but a small part of his fruitful activity in technical literature. Another part is comprised in the two series of articles (more than fifty in all) published at

intervals during about fourteen years in *The American Engineer and Railroad Journal*, and intended to explain and recommend to manufacturers the specifications of the Pennsylvania Railroad Co., by stating the reason of such requirements, and describing the methods of the chemical tests involved. And to these must be added his numerous papers and addresses before technical societies, etc., of which I have been able to collect the following incomplete but suggestive list:

Fuel Oil. Lecture delivered before the Franklin Institute, Jan. 6, 1888.

International Standards for the Analysis of Iron and Steel. Reports of the Subcommittee on Methods, Charles B. Dudley, Chairman. (*Proceedings of the American Society of Civil Engineers*, 1889 and 1890.)

Lecture on Paints and Painting Materials. Delivered under the auspices of the Carriage Builders' National Association, Mar. 16, 1892.

Address to the Members of the Chemical Section of the Engineers' Society at Pittsburg, Sept. 27, 1892.

Standard Methods for the Analysis of Iron and Steel. Read before the Engineers' Society of Western Pennsylvania, Oct. 24, 1893.

An Attempt to Find the Amount of Phosphorus in Three Samples of Steel, and *Some Points in the Determination of Phosphorus in Steel by the Volumetric Method.* By Charles B. Dudley and F. N. Pease. *Journal of the American Chemical Society*, Vol XVI., No. 4, April, 1894.

The Theory and Practical Use of Inert Pigments. "*Drugs, Oils, and Paints*," October and November, 1896.

Some Present Possibilities in the Analysis of Iron and Steel. Read before the American Chemical Society, Dec. 29, 1896.

The Ventilation of Passenger Cars on Railroads. *Journal of the Franklin Institute*, July, 1897.

The Dignity of Analytical Work. Presidential Address. Read before the American Chemical Society, Dec. 29, 1897.

Report of the Committee on Coal Analysis. By William A. Noyes, W. F. Hillebrand, and C. B. Dudley. *Journal of the American Chemical Society*, Vol. XXI., No. 12, December, 1899.

Disinfection of Passenger Cars. By C. B. Dudley, Chemist, and M. E. McDonnell, Bacteriologist, of the Pennsylvania Railroad Co. *American Engineer and Railroad Journal*, June, 1902.

The Making of Specifications for Materials. President's Address. *Proceedings of the American Society for Testing Materials*, Vol. III., 1903.

A System of Passenger-Car Ventilation. Read April 8, 1904.

Oils and Pigments. Lecture before the Sixteenth Annual Convention of the Master House-Painters' and Decorators' Association of Pennsylvania, Jan. 14, 1904.

The Passenger-Car Ventilation System of the Pennsylvania Railroad Company, 1904.

The Influence of Specifications on Commercial Products. President's Address, American Society for Testing Materials, June, 1904. *Proceedings*, Vol. IV., 1904.

The Testing Engineer. President's Address, American Society for Testing Materials. *Proceedings*, Vol. V., 1905.

The Dissemination of Tuberculosis as Affected by Railway Travel. Read at the annual meeting of the American Public Health Association, September, 1905.

The Enforcement of Specifications. President's Address, American Society for Testing Materials. *Proceedings*, Vol. VII., 1907.

Some Features of the Present Steel Rail Question. President's Address, American Society for Testing Materials. *Proceedings*, Vol. VIII., 1908.

Engineering Responsibility. President's Address, American Society for Testing Materials. *Proceedings*, Vol. IX., 1909.

This wonderful aggregate constituted one of the most potent factors in that harmonious union of practitioners with critics and investigators which has produced the rapid progress of the arts as well as the sciences among us. Each of these classes has reacted as a stimulus upon the other; and Dudley represented both.

He was a member of the American Societies of Civil and Mechanical Engineers; the American Institute of Electrical Engineers; the American Electro-Chemical Society; the Franklin Institute; the American Philosophical Society; the Academy of Science; the American Railway, Historical, Forestry, and Public Health Associations; the Iron and Steel Institute of Great Britain; the Verein deutscher Eisenhüttenleute; and numerous English, German, and French chemical societies. He was twice President of the American Chemical Society, and four times of the American Society for Testing Materials; President of the Bureau of the American Railway Association for the Safe Transportation of Hazardous Materials; and Chairman of the Committee on Explosives of that association. At the time of his death, he was President of the International Association for Testing Materials. He was one of the founders of the Juniata Club in Altoona, and a member of the Cosmos (Washington, D. C.), Union League (Philadelphia), and Engineers' (New York) clubs.

The town of Altoona, which his work helped to make famous throughout the world, owes him a great debt of gratitude. He was the President of its first Board of Health, and placed at its service his scientific knowledge and business sense in the establishment of good sanitary conditions. The Pennsylvania Company, which owns so large a part of the town and employs so large a part of its population, had given him its complete confidence, and was accustomed to "do what Dudley said;" and his unselfish labors won for him likewise the confidence of his townsmen; so that he had, in many respects, the power of

a dictator. Yet he was but "the servant of all," and only in that best sense a ruler. One of the institutions which bears the marks of his helpful guidance is the Altoona Mechanics' Library, of which he was an officer, and for which, aided by the Pennsylvania Railroad Co., he acquired many well-selected technical books and files of scientific periodicals. The library now contains more than 45,000 volumes, comprising, besides adequate provision in general literature, history, travel, and fiction, a collection of standard books of reference in science and technology which many a college might envy.

But it was not as a kindly patron from some larger sphere, bestowing benefits incidentally, as it were, upon a community in which he had only a benevolent interest, that Dr. Dudley served the town of Altoona. He regarded it as his home; he invested in it his savings; and on its eastern edge he had a farm, to which, rather than to fashionable watering-places or foreign lands, he retired for rest and recreation. He had made four journeys to Europe, having been sent in 1886 to study, in Russia especially, the use of oil as a fuel in railway-locomotives, and again in 1900 as a delegate to the International Railway Congress at Paris. In 1906, he spent a happy three months' vacation in Italy, with his wife; and in 1909 he made another transatlantic trip. But round the world, as here among us, he was known and hailed as "Dudley of Altoona." The journey of 1909 was made for the purpose of attending as a delegate from the United States the Copenhagen meeting of the International Society for Testing Materials. It was at this meeting that he was enthusiastically elected President of the Society by the 900 delegates present from all parts of the world. This honor touched him deeply, and his satisfaction was increased by the selection of the United States, upon his invitation, as the country in which the next international meeting should be held. In extending this invitation, Dr. Dudley promised that the sessions should be carried on in English, French, and German. He felt that the proper execution of this promise involved the use of these languages by the presiding officer, and he was conscious of his own inability, through lack of practice, to speak freely in French or German, although he could read both languages with ease. It was characteristic of him, that immediately after the meeting at Copenhagen he began

to take lessons in talking French and German, and devoted to that purpose every available hour of his time thereafter. It is not easy for a man approaching seventy years to acquire facility in a new tongue; but I firmly believe that Dr. Dudley would have conquered this difficulty before 1912.

The story of his final illness and death is brief. Up to the middle of December, 1909, he was, as we were all accustomed to see him, full of vitality and unimpaired in his enthusiasm and power for work. But on Friday, Dec. 17, he returned to Altoona from a two days' visit to Washington, D. C., where he had contracted a severe cold. The next day pneumonia was developed; on Sunday his condition became alarming; and on Tuesday evening he died. We are told on high authority that pneumonia is one of the very few maladies in dealing with which modern medical science has made little or no progress. It seems a cruel mockery of our human knowledge that this one unconquered foe should so easily overcome a great scientific leader. Yet, if we contemplate the character of the man, and the work he had already accomplished for his kind, we may rather lament our own continued careers than that which was so suddenly cut off. If, using the method of engineers, we should estimate the value of a life by multiplying its years into its achievements, who of us might not envy this record? And if, in similar fashion, we take account of its momentum, as it disappears from our mortal view, who of us will dare to say it has ceased to advance, because we have ceased to see it? Is anything ended, except our power of seeing?

I cannot do better, in summarizing both the qualities of Charles B. Dudley and the impressions which they produced upon others, than to quote the following paragraphs from the minute of the Executive Committee of the American Society for Testing Materials, prepared by Prof. Henry M. Howe, past-President of that Society, of which Dr. Dudley himself was President at the time of his death, and Dudley's successor as the President of the corresponding International Society:

"Let us record at once our deep grief and our deeper gratitude, our grief indeed at the loss of a great leader and dear friend, but above all, our gratitude that we have had the privilege of being led.

"Such measure of usefulness as our Society has had, it owes in very large part to that leadership. Here was a most rare combination of qualities, the sterling,

the intellectual, the human, the judicial, each on a high level, all combining to form a character, a personality, whose like we shall not look upon again.

"With a clear head to see the world's needs, to part the essential from the accidental and the merely concomitant, went the skillful and persuasive tongue to make clear to the rest of us what he had first made so clear to himself. With these went the perfect fearlessness, apparently even the unconsciousness of either danger or fear, which made him lead on where others would have flinched. With these again went his calm, clear, good judgment, which seemed to tell him spontaneously which among the good things that needed doing were the most worthy of being done, and what were the best and surest ways of doing them.

"With all these admirable qualities went that which was necessary to the accomplishment of his high purposes, his kindness of heart, his sympathy and his tact, which made us all his allies in what he undertook. Had he a proposal? Our affection and veneration for him made us almost its advocates before it was unfolded. Its intrinsic wisdom, and the clearness with which he expressed it, found an audience ready, almost anxious, and certainly expecting to be convinced."

Though keenly aware of the inadequacy of my notice of this great leader, bright example, and dear friend, I cannot believe that I have wholly failed to set forth beyond misunderstanding the features which seem to me most important and inspiring in his career—namely:

1. He was among the first to furnish in this country a conclusive demonstration of the value of scientific aid in business enterprises.

2. The demonstration which he thus furnished proceeded from his own versatile and sympathetic character, and consisted in the tactful combination of practical application with theoretical research.

3. The extraordinary effect of his labors in this field was enhanced by the fraternal spirit in which he shared their results with other workers in the same field.

4. And supremely important, in his work, was his personality; the unfailing sweetness, cordiality, and victorious joy of his temperament. He loved the world; he loved his fellowmen; and we loved, trusted, and followed him!

I have reserved to the last an observation for which I could find no appropriate place in the above text, yet which seems to me essential to a full view of Dr. Dudley's life.

We who were personally intimate with Dr. Dudley had become accustomed to regard him as one of those "bachelors of science" who, though they had long ago become "Masters"

and "Doctors," still held to the celibacy connected, in old times, with their first degree. And when we learned of his marriage, Apr. 17, 1906, to Miss Mary V. Crawford, of Bryn Mawr, Pa., we rejoiced that, even so late, he had found the home-life which he so well deserved, and for which he was so well qualified. Our hopes and congratulations were not disappointed. We saw how that experience renewed his youth; we read upon his face the one last word of charm; we recognized the flying signal of a heart, anchored at last in a happy home. And, even in this public place, we would offer, together with our praise of him, both our sympathy and our congratulations to her whose affection added flowers of crowning joy to the laurel-leaves of his fame!

Biographical Notice of William Phipps Blake.*

BY ROSSITER W. RAYMOND, NEW YORK, N. Y.

(Canal Zone Meeting, November, 1910.)

THE death of Professor Blake removes the oldest of American economic geologists and mining engineers, and deprives this Institute of one of its earliest and most illustrious members. To many of us it brings a personal loss also, the loss of a friend.

William Phipps Blake was born June 21, 1826, in New York City. His father, Elihu Blake, was a surgeon-dentist of eminence, a direct descendant of William Blake, who settled in the Massachusetts Bay Colony about 1630. He was prepared for college at private schools in New York, and entered the Sheffield School of Yale, where he was graduated as Ph.B. in the chemical course, class of 1852, the first graduating-class of the institution. In the same year he became chemist and mineralogist of the New Jersey Zinc Co., and chemist of chemical works at Baltimore, Md. In 1853, he started the Department of Mineralogy of the World's Fair at New York City. In 1854, 1855 and 1856, he was mineralogist and geologist of the United States Pacific Railroad Surveys of Williamson, Pope, and Whipple, and for the War Department at

* See Frontispiece of this volume.

Washington. His writings during this period comprise reports on the geology and mineralogy of California and other parts of the Southwest, and constitute well nigh the earliest scientific accounts of the regions described. One of them was a translation of the Résumé and Field Notes of Jules Marcou, of Whipple's expedition. From 1856 to 1859, he was engaged in explorations of the geology and mineral deposits of North Carolina, and other parts of the country. In 1859, he became editor and proprietor of *The Mining Magazine*. This was a monthly periodical, founded in 1853 by W. J. Tenny, who conducted it until 1858, when it passed into the hands of Thomas McElrath, a New York publisher, and appeared in May of that year as *The Mining and Statistic Magazine*, with the name of Robert G. Rankin as "scientific collaborator" upon the cover. In 1859, Mr. McElrath disposed of the magazine to Blake, who transferred the publication to New Haven, and issued the number for November, 1859, as a "second series" of the original periodical, under the title *The Mining Magazine and Journal of Geology, Mineralogy, Metallurgy, Chemistry and the Arts*, in their Application to Mining and Working Useful Ores and Metals, Edited by William P. Blake, geologist and mining engineer. It will be noted that the wide field thus claimed by him for his journalistic enterprise was almost exactly that which the American Institute of Mining Engineers occupied, twelve years later.*

The pages of the *Mining Magazine*, under his editorship contain much information of value, and exhibit the promise of an important and influential future. But the time was unpropitious for such an enterprise. The approach of war hindered the further investment of capital in the Southern States, the mineral resources of which Blake had recently explored and was prepared to describe. The far West was too far away to support, in the absence of railroads, a periodical which could not be cheaply and surely transported to its subscribers. In short, the new series of the *Mining Magazine* had to be sus-

* In his interesting paper, Notes and Recollections Concerning the Mineral Resources of Northern Georgia and Western North Carolina (*Trans.*, xxv., 796), presented at the Atlanta meeting of 1895, Professor Blake quotes from *The Mining Magazine, The Mining and Statistic Magazine*, and *The Mining Magazine and Journal of Geology*, thus indicating the successive stages of its history.

pended in 1860; and its editor accepted an engagement which prevented him from reissuing it, namely, that of mining engineer to the Japanese government. This position he occupied from 1861 to 1863. In company with Raphael Pumpelly, he organized the first school of science in Japan, and taught chemistry and geology in the school and in the field. He visited China also, returning to the United States by way of Russian America (Alaska). At this time (1863) he explored the Stickeen river, discovered the great Stickeen glacier, and wrote the first description of what was then supposed to be "the ice-mountains." It is said that his reports to Secretary Seward were influential in procuring the consummation of the purchase of Alaska.

Upon his return to California in 1863, he resumed his work as a field-geologist and mining expert, studying especially the character and development of the Comstock lode. In 1864, he was appointed Professor of Mineralogy and Geology in the College of California, and also mineralogist of the State Board of Agriculture. In 1867, he was appointed Commissioner for California to the Paris Exposition, and was selected by the State Department to edit the Report of the United States Commissioners. This work, which embraced six volumes, occupied him mainly until 1871. But he found time in 1869 to preface an account of the mechanical appliances of mining which formed a part of my report as Commissioner of Mining Statistics.

In 1871, he was selected as chief of the scientific corps of the United States expedition to San Domingo, and led his party across that island. In 1873, he was appointed a Commissioner of the United States to the Vienna International Exposition, for the report of which he wrote the portion devoted to iron and steel.

His efficient service in connection with two International Expositions led to his appointment by the Smithsonian Institution in connection with the collection and installation of the United States exhibit of mineral resources at the Centennial Exposition, in 1876. This collection formed the nucleus of the Mineral Department of the National Museum at Washington.

At the Paris Exposition of 1878, he was again one of the

United States Commissioners, and was appointed Secretary of the scientific part of the Commission. Besides editing the report, he wrote several of its chapters, on Ceramics, Glass, etc.

For the following 15 years or more, Professor Blake was actively and widely engaged as an economic geologist in the exploration of districts and the examination of mines in Arizona, California, Utah, Nevada, Idaho, Montana, and other States and Territories, and published many articles in technical periodicals. A glance at the Appendix to this paper will show the extraordinary range of his work. In 1895, he was appointed Professor of Geology and Mining, and Director of the School of Mines, at the University of Arizona, Tucson, Ariz.; and although he was already in his seventieth year, he engaged in this new enterprise with all the ardor of youth, and prosecuted it with vigor and success for 10 years. In 1905, he resigned his position, becoming Professor *emeritus*. In January, 1898, he was appointed by the Governor of Arizona, Territorial Mineralogist and Geologist—an office to which no salary was attached, but which he accepted with generous public spirit, and the duties of which he discharged until his death 12 years later.

Professor Blake received the honorary degree of M.A. from Dartmouth College in 1863, that of Sc.D. from the University of Pennsylvania in 1906; and that of LL.D. from the University of California in 1910. He was made a Chevalier of the Legion of Honor of France in 1878.

It was while in Berkeley, Cal., to attend the semi-centennial anniversary of the University of California, that Professor Blake died. For some years past, he had been accustomed to spend his winters at Tucson, returning to his home at Mill Rock, New Haven, Conn., for the summer months. This year he was invited as one of the earlier professors of the College of California to attend, as the guest of the University, the semi-centennial celebration at Berkeley, and to receive, in recognition of his "distinguished services to geological science," the honorary degree of Doctor of Laws. He left Tucson May 12 (on his return from a geological field-examination in Arizona) and reached Berkeley May 14, after a fatiguing journey. Instead of resting, he fulfilled with indomitable energy several social engagements already made, before yielding to physical

weakness and taking to his bed. Even then, he could not submit to be treated as an invalid. In spite of the urgent warning of his physician, he arose, dressed, and appeared in cap and gown at the Greek theater, on Wednesday, May 18, to receive his degree. From this academic triumph, he returned to his bed, which he was not to leave again. Pneumonia was rapidly developed; and he died peacefully and in full possession of consciousness early on Sunday morning, May 22, 1910. It was the happy end of a long, honorable, laborious and useful career.

Professor Blake became a member of this Institute at its first meeting, in May, 1871, and was elected a Vice-President immediately. In 1872, 1873, and 1874, he was unanimously re-elected; in 1875 (the new rules, adopted in 1873, having limited the continuous term of a Vice-President to two years), he could not be re-elected; but in 1876 he was restored to the position of Vice-President, and served until 1878. Twenty-seven years later, as a veteran of 79, he received once more the honor of a Vice-Presidency; and I remember well the example of fidelity which he set for younger men, by his attendance at the meetings of the Council, to which he came from New Haven.

His contributions to the *Transactions* were numerous and valuable, as the Appendix to this notice abundantly proves. One of the most important of them deserves particular mention here.

In 1902, Henry T. Blake, of New Haven, a son of Eli Whitney Blake, the inventor of the Blake stone-breaker, for himself and the other heirs of that distinguished pioneer, established a fund of six hundred dollars, to be administered by the Board of Trustees of the Sheffield Scientific School of Yale University, for the purpose of periodically awarding prizes to the authors of worthy treatises upon subjects of mining or civil engineering, and especially in some branch involving the use of broken stone or ores. The trust-deed, stating the conditions of this trust, is given in full in our *Transactions* (vol. xxxiii., page l). The first award of this prize was made at the New Haven meeting of October, 1902, when it was given to William P. Blake for an admirably comprehensive and suggestive paper on The Blake Stone- and Ore-Breaker, Its Invention, Forms,

and Modifications, and Its Importance in Engineering. The pertinency of the subject, the ability of the paper itself, and the fact that the author was a nephew of the great inventor, and a member of the first graduating-class of the Sheffield School, combined to make this first award of the prize a singularly just and happy act.

I first made the acquaintance of Professor Blake in 1868, when I was beginning my work as United States Commissioner of Mining Statistics. In that work I received from him invaluable assistance; and our acquaintance ripened to a friendship which was never broken. In 1873, we were both Commissioners to the Vienna Exposition; and after we left Vienna, he and his charming wife * joined our party in a journey by carriage through the Bavarian Tyrol. They were ideal traveling-companions, merrily superior to all inconveniences of the way, and eagerly appreciative of scenic beauty, historic associations, nature, and human nature.

The portrait accompanying this notice will recall to many friends the striking personal beauty of Professor Blake. His hair turned white while he was still a young man, and retained throughout his life its abundant growth, which, together with his noble face, gave to his head almost the aspect of the Phidian Jove. But the clear, ruddy complexion, bright eyes, and genial smile made him too sympathetically human for such a comparison. In conversation, he was fascinating, by reason of his own keen interest in what he was saying. He told a fact as if he had only just discovered it. In the art of delivering in "oral abstract" the substance of a technical paper, and illustrating his remarks by rapid black-board sketches, he had no superior. He did such things with the grace, directness, and lucidity of a generation not pampered with stenographers, type-writers, and lantern-slides. Out of our earthly life he has departed—stalwart, versatile, tireless, brave, and gentle to the last;—but from my soul, at least, his spendid presence and his serene yet eager spirit will never depart.

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* Professor Blake married, Dec. 25, 1855, Miss Charlotte Haven Lord Hayes, a daughter of William Allen Hayes, of South Berwick, Me. She died in 1905, at Mill Rock, New Haven, in the fiftieth year of happy wedlock. Of their six children three survive them: Francis H. Blake, Litchfield, Conn., and Joseph A. Blake, M.D., and Theodore Whitney Blake, of New York City.

Appendix.

The following summary of Professor Blake's work is reprinted from the President's Report to the Board of Regents of the University of Arizona for 1909. To the list here given must now be added what was probably his last contribution to technical literature, the brief paper on Manganese-Ore in Unusual Form, which will be found in this volume, page 647.

INTRODUCTION.

For nearly sixty years William Phipps Blake has been a contributor to the literature of science. Even before his graduation in 1852 from the Sheffield Scientific School of Yale University, in its first class, he was writing for the *American Journal of Science*. The wide range of his observations, and the great diversity of titles in this list of his writings are no less remarkable than the span of decades which they cover. The United States—especially Arizona, California, Utah, and Wisconsin—Mexico, Alaska, England and Japan are debtor to his keen eyes, his penetrating understanding, and his prolonged investigations. His is no merely scribal pen; it writes with authority. This bibliography lays no claim to completeness, full as it is, for no effort has been made to trace out all contributions to the weekly and daily press. The University is very glad, however, to be able to collect and present in permanent form, on the sixtieth anniversary of the first published article, this notable list of the writings of a man who has been for so long a devoted friend of the University and of Arizona, and for two generations a distinguished explorer in geology, mineralogy, and mining engineering.

Kendric Charles Babcock.

March 10, 1910.

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* Arranged chronologically, save for contributions to the chief periodicals and proceedings of learned societies, which are grouped together.

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Biographical Notice of William Metcalf.

BY R. W. RAYMOND, NEW YORK, N. Y.

(Pittsburg Meeting, March, 1910.)

AT the Pittsburg meeting of the Institute, in March, 1910, the death of Mr. Metcalf was announced, and Col. H. P. Bope, of Pittsburg, delivered in memory of him a brief but eloquent address, which, though expressing the sorrow of Mr. Metcalf's friends, and, in general terms, the debt of gratitude which his country and his profession owed to him, was not intended to be a complete survey of his life and work. With Colonel Bope's permission, and with the aid of some of the statements in his address, I substitute for it, in our *Transactions*, this more detailed, yet still inadequate memorial. As Colonel Bope justly remarked, the men who could do justice to this theme—such men as Jones, Chalfant, Bennett, Oliver, and Park—passed away before Metcalf, leaving him almost the last of the generation of great steel-makers who made Pittsburg the Sheffield of America, and who discharged a double function, as the rear-guard of the old metallurgical practice and the van-guard of the new. I knew many of them; would that I were better qualified to celebrate their achievements.

William Metcalf was born Sept. 3, 1838, at Pittsburg, Pa., where his father, Orlando Metcalf, was an eminent member of the bar. At the age of sixteen, he entered the Rensselaer Polytechnic Institute, at Troy, N. Y., from which he was graduated in June, 1858. Returning to his native city, he became an assistant engineer in the Fort Pitt Foundry, of which he was made Superintendent in 1859. This famous establishment produced the largest castings and heaviest machinery then known in the United States; and Mr. Metcalf's intense, intelligent and incessant study of his business prepared him as a manufacturer for a patriotic service to his country more valuable than, as a soldier, he could have rendered in the field. The call came in 1862, when the government needed, above all things, a supply of field, naval and siege artillery. The

services of Abram S. Hewitt and Alexander L. Holley in this line are well known.[1] With their names, that of Metcalf should stand, perhaps as foremost of the three; for he had sole charge of the manufacture at the Fort Pitt foundry, during the war, of guns and projectiles, and made more than 3,000 pieces of artillery, comprising 8-, 10-, 13-, 15- and 20-in. guns for the army; 9-, 11-, and 15-in. guns for the navy, and 8-, 10-, and 13-in. mortars. In this list belong thirty 13-in. mortars which were finished and shipped to General Grant within 40 days after receipt of the order, and which threw into Vicksburg, Miss., during the siege of that place, more than 1,000 tons of shells: also, seventy-five 4-in. "Rodman rifles," which did good service at the siege of Petersburg, Va.

I take the foregoing particulars from a memorandum furnished by Mr. Metcalf at my request in 1903, which I had not seen since, until it was unearthed for present use. That it had not been mislaid or forgotten, I count most fortunate—not only because the figures it contains constitute of themselves a significant record, but also, and chiefly, because Mr. Metcalf added to his statement of them the simple but, to me, tremendous sentence:

"Not one gun of Fort Pitt make was ever reported as failing in service."

That sentence, penned by his own hand, and lying before me as I write these lines, seems to me to constitute a comprehensive and appropriate epitaph of its deceased author, who, like the multitudinous productions of his skill and care, never failed in service!

Late in 1867, Mr. Metcalf gave up the foundry business, and in January, 1868, joined the firm of Miller, Barr & Parkin, (later, Miller, Metcalf & Parkin, and finally the Crescent Steel Co.), of Pittsburg, manufacturers of fine crucible steels, under the "Crescent" trade-mark. To finish the recital of his business enterprises, I may add here, that he retained his connection with this company until the end of 1894, and that, early in 1897, he established, at Braeburn, Pa., the Braeburn Steel Co., with which, I believe, he was connected to the end of his active career.

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[1] See my Biographical Notice of Abram S. Hewitt, *Trans.*, xxxiv., 186; my address in the *Memorial of Alexander L. Holley*, and Holley's *Ordnance and Armor*.

It was during his management of the Crescent Steel Works that Mr. Metcalf became most widely known as an authority on the qualities and manufacture of steel. He became a member of this Institute in 1875, while it was, though rapidly growing, still numerically feeble. As our *Transactions* of that period abundantly show, the metallurgy of steel was one of our "burning questions"; and I well remember the position and influence of Mr. Metcalf, as one of the first practical experts to emphasize the importance of mechanical treatment and heat-treatment, as compared with chemical composition, and also the different effects of different kinds of tests of strength. Master of all the practical features of his trade, he showed us, at his own works, how easily a bar of steel could be broken in one way, though in no other; and his shrewd, keen criticisms tempered greatly the zeal of the ardent theorists among us. A part only of his valuable advice is contained in the list of his contributions, given below. The rest will be found in his other publications, or else, like so much of the precious wisdom colloquially imparted by veteran practitioners, will never be found at all, in connection with his name—though somebody who first learned it from him may have acquired credit galore by making it known.

Contributions of William Metcalf to the Transactions of the Institute.

Papers.

Title.	Vol.	Page.	Year.
Can the Commercial Nomenclature of Iron be Reconciled to the Scientific Definitions of the Terms Used to Distinguish the Various Classes?	V.	355	1876
Can the Magnetism of Iron and Steel be Used to Determine Their Physical Properties? . .	IX.	385	1881
Natural Gas, ...	XIV.	589	1886

Discussions or Remarks.

Chimneys for Siemens Heating Furnaces, . .	IV.	108	1875
Economy of Fuel in Siemens Producers, . .	V.	429	1877
The Nomenclature of Iron, ...	V.	532	1877
Papers on Steel Rails, ...	VII.	379	1879
Papers on Steel Rails, ...	IX.	547	1881
Iron and Steel Considered as Structural Materials,	X.	403	1882
The Condition and Action of Carbon on Steel, .	XXXIV.	979	1904

Mr. Metcalf's cordial and effective co-operation in the work of the Institute won quick recognition. He was elected a

Vice-President for 1878–79, and President in 1881. Other professional societies followed our example. He became President of the American Society of Civil Engineers in 1893, President of the Engineers' Society of Western Pennsylvania in 1880, Vice-President of the American Society of Mechanical Engineers in 1882. To these societies he contributed numerous professional papers on various subjects connected with the manufacture and qualities of steel. He was also a member of the Institution of Civil Engineers, of Great Britain, and a Fellow of the American Association for the Advancement of Science.

In 1897, he published a little book, entitled "Steel: a Manual for Steel-Users," which took rank at once as a practical classic, if I may use that term. The theory of the composition and nature of steel has made great progress since that date; Mr. Metcalf's explanations may have been, in this or that point, superseded; but nothing can supersede the facts he knew and told. His work was used as a text-book of instruction in several technical schools; and, even now, all students can read it with advantage.

As a citizen, Mr. Metcalf enjoyed the complete confidence of the people of Pittsburg, of which a striking instance was mentioned by Colonel Bope, in his address at our Pittsburg meeting of March, 1910, mentioned at the beginning of this notice. Namely, Mr. Metcalf was one of the Board appointed to appraise the property of the Monongahela Navigation Co., which the U. S. government desired to buy. The Board awarded to that company the sum of $3,000,000; and its decision was accepted without cavil or criticism by both parties and by the public, as a just and upright verdict.

Mr. Metcalf filled also many positions of trust, the duties of which he discharged to universal satisfaction.

Mr. Metcalf died in Pittsburg, Dec. 5, 1909, at the age of 71. His active, useful, yet quiet and modest life; his upright and amiable personal character; the unstained reputation which he has left behind him—these things warrant me in saying once more what he wrote me of his Fort Pitt guns: He never failed in service. Could any man have higher praise?

Biographical Notice of Franklin R. Carpenter.

BY H. O. HOFMAN, BOSTON, MASS.

(Canal Zone Meeting, November, 1910.)

THE sudden decease, April 1, 1910, in Chicago, of Dr. Franklin R. Carpenter was a shock to his many friends. He died in his sixty-second year, of heart paralysis. To most fellow-members of the Institute who have followed the progress of the last twenty-five years in the mining and treatment of non-ferrous ores, the work of Dr. Carpenter in that line is familiar; but very few, even of those who knew him well in the later years, are acquainted with his earlier struggles to satisfy an inborn hunger for knowledge. The story of his life resembles that of the pioneer who opens up new country; overcoming hardships with indomitable courage, meeting doubts and difficulties with an alert and open mind, and exercising in the face of reverses an unconquerable perseverance. It seems appropriate to place on record in the *Transactions* a brief outline of the life and work of this eminent engineer.

During the past 15 years of his life Dr. Carpenter was much interested in genealogy. In the pursuit of this study as an avocation, he traced his own ancestry to the time when "Hugh," called "le Charpentier" on account of the dexterity with which he had swung his battle-axe as a crusader, settled in Sussex on lands received from William the Conqueror. The Carpenters came to this country with William Penn. A prominent member of the family was Joshua Carpenter, the founder of Christ Church, Philadelphia. Dr. Carpenter's branch of the family moved from Pennsylvania into that part of Virginia which is now West Virginia, where his grandfather settled in Harrison county, while the grand-uncle went further west, and built the first iron-furnace at Hanging Rock, Lawrence county, Ohio. In fact, since early times, the Carpenters appear to have been iron-masters.

Franklin R. Carpenter was born, Nov. 5, 1848, in Parkers-

burg, W. Va. When he was less then four years old, his father died, leaving a widow and three children, of whom Franklin was the eldest. As the widow did not have abundant means, she went about 1853 from Parkersburg to Clarksburg, in Harrison county, to teach in the Broaddus Seminary. Later, she became postmistress; her eldest son, at the age of seven or eight, handling the mail bags. During the Civil War the now 12-year-old boy was frequently sent through the lines at Clarksburg with dispatches sewed up in his clothes. In his 14th year he became a "boy" in a Parkersburg hardware-store; in his 16th year he was apprenticed to a jeweler and watch-maker. Although skillful at his trade, he was not content to surrender his ambition for a higher education. He read everything that came within his reach. Returning to Clarksburg in 1864, he succesfully passed the examination for a teacher's certificate, and alternated between school-teaching and farm-work. In 1865 he studied Latin, mathematics, and natural philosophy, and read much under the guidance of the kindly village physician, Dr. Late; so that he was able to earn a teacher's certificate of the highest grade, and to command a better salary.

The survey of the Baltimore & Ohio R. R. in that section formed the turning-point in his career, by arousing his desire to become a civil engineer. With the money saved while teaching, he entered in 1866 a small denominational school at Pruntytown, W. Va., called Rector's College, and took the course in civil engineering. His proficiency in mathematics won him the friendship of his teachers. He was graduated in 1868, and had his first engineering experience as rodman with a railroad-surveying party at Chillicothe, Mo. He was quickly advanced to transit-man, and after six weeks became chief of the party. But being attacked by malarial fever, he returned in 1869 to West Virginia, and became Assistant Geologist in the State engineer corps. His early experience in laying out railroads doubtless developed the characteristic topographical instinct in geological field-work which enabled him to locate quickly the critical points for observation.

In 1872, he invested his savings in his first financial enterprise, by bonding timber- and coal-lands in West Virginia; but, like most first ventures, this one came to naught; and he was

obliged to betake himself to school-teaching—this time at Fetterman, W. Va. His engagement in 1873 to Miss Annette Howe, of Athens, Ohio, became the incentive to another westward journey in search of better fortune. He went to Kansas, and later to Denver, Colo., to make a new start in railroad-work. But the panic of 1873 paralyzed all such enterprise, and the young adventurer was reduced to the earning of a bare living by washing gold on Spanish Bar, at Idaho Springs, Colo. There, his character and ability were soon recognized; and he was elected mayor and police-judge, and afterwards taught school until the fall of 1874, when he removed to Georgetown, where, receiving the mark of 100 in a competitive examination, he obtained the office of Principal of Schools. On the strength of this achievement he was married, Dec. 23, 1874. In 1876 he was elected County Superintendent of Schools, and retained this position until 1878, when he opened in Georgetown an office, and rapidly built up a good practice as U. S. deputy surveyor and civil engineer. Two pieces of his engineering work are worthy of record. In 1881 he made for Captain Berthoud the first survey of the "loop" above Georgetown; and in 1882 he located the Loveland Pass tunnel and started work at both portals. In 1882, he took a lease on the Corry City mine with the intention of driving a cross-cut tunnel to tap the lode; but his financial backing gave out, and the scheme had to be abandoned to others, who brought it afterwards to a successful issue.

For a while, he returned to his native State, West Virginia. But having tasted Western freedom and enterprise, he could not be held long in the East. In 1885 he started for British Columbia, but was taken ill in Minnesota and forced to return. In 1886, however, attracted by the new tin-deposits of the Black Hills, he went to Rapid City, S. D. His leading work that year was the location, through the western part of the Black Hills, of a railroad line, which is now occupied by a part of the Burlington & Missouri R. R. At the same time, he made geological observations which were helpful to him later.

In December of the same year he was elected Dean of the new Territorial School of Mines, which he opened in 1887, occupying the chair of geology and mining. The Burlington & Missouri R. R. Co. recognized his ability, and made him its

mining engineer—a position which he held until his death. In June, 1887, Ohio University conferred upon him the degree of Master of Arts, and in 1888 that of Doctor of Philosophy. In 1889 he became Fellow of the Geological Society of America, and later Fellow of the American Association for the Advancement of Science. His early reconnoissance of the geology of the Black Hills prompted the Territorial government to make an appropriation for a geological survey, which was started in 1887, and finished in 1888.

In 1887 I was called to the Dakota School of Mines as Professor of Metallurgy and Assaying, and there made the acquaintance of the man who was to become my best friend. The beginning of Dr. Carpenter's life, as indicated, was by no means prosperous. Although he came on both sides from old and honored families of Pennsylvania and Virginia, his parents were in reduced circumstances when he was born, and his prospects for the future were not promising. Starting as the boy in a hardware-store with the few rudiments of elementary schooling, he acquired by personal effort without outside assistance the knowledge to become a Master of Arts of the oldest university west of the Ohio river, and later to prepare a geological thesis of sufficient value to deserve the degree of Doctor of Philosophy. In his engineering study and practice he was as progressive as in his purely scientific attainments. In general culture he had made himself familiar with the thought of the leading writers in the English language, in fact had written an epic poem, "The Quest of Saint Brendan," well spoken of by critics of literature. And this was in 1887, when he had still 23 years ahead of him, which he did not allow to lie fallow. Truly a result which can be attained only by a man of talent and with persistent effort. All his acts showed him to be a man of kindness of heart, of loyalty to friend and principle, generous to a fault, punctilious when he had charge of other men's affairs, easy-going with his own. A striking characteristic was his devotion to wife and family; he lived for his home, the sons found in him their best friend. He was a devout Episcopalian, having become an active member of the church at the early age of 15; his poetic nature made it natural for him to favor the high-church branch of the denomination.

His ideas of right and wrong were well brought out in his management of the Dakota School of Mines. It is an accepted saying that Territorial or State schools pass through a political and a denominational stage before they reach the correct basis for future growth. In the first stage, the expenditure of moneys for buildings and apparatus is likely to involve political patronage; and when this period is over, and the politicians are no longer actively interested, a denominational stage is likely to follow, in which "good men" who have not been successful are provided with berths as "educators." At last the community gets tired of the state of affairs; there is a change of *personnel*; and the institution is set upon the true road to usefulness and success. In 1887 the Dakota School of Mines was in the political stage. The Dean had to hold the fort against all open assaults, as well as cunning stratagems, of the politicians. He kept flying, high and unstained, the flag of principle, and never lowered it at any command of interest. What this means can be fully understood, perhaps, by those only who have gone through a similar experience.

When I was called, in 1889, to another educational institution, and my place was to be filled by the appointment of a successor whom the Dean conscientiously disapproved, he resigned his office without hesitation, although he had at the time no certain prospect of remunerative employment elsewhere.

His genius soon found, however, a new path. He had tested, by crucible-experiments in the laboratory, the possibility of melting the siliceous silver-gold ores of Ruby Basin, S. D., and had found that this could be done so as to obtain a satisfactory recovery of the precious metals. What remained to be determined was the kind of slag that could be economically produced by this process in a blast-furnace. This he undertook to determine at Deadwood in a small blast-furnace ("baby-smelter"); and when this small operation had proved successful, he laid the foundations, in mining-property and plant, for a smelting enterprise on a commercial scale. In 1891 he erected the Deadwood and Delaware smelter, which, by 1898, had grown to comprise five blast-furnaces. During this interval, he carried on many experiments in semi-pyritic smelting, and developed a new mode of operation in this interesting branch of metallurgy. This smelter, destroyed by fire

in 1898, was rebuilt by Dr. Carpenter and managed by him until the property changed hands, a year or two later.

Domestic afflictions and the impaired health of his wife led him to leave the Black Hills in 1900 and return to Denver, which remained thereafter his place of residence. His first new work in Colorado was to straighten out the difficulties of the Buena Vista plant; in the same year he designed and built the works of the Rocky Mountain Smelting Co., at Florence, and in 1901 he did the same kind of work for the Clear Creek Mining & Reduction Co., at Golden, the plant of which he managed until 1903. In 1904 he turned his attention to the electrostatic concentration of the ores of the Nonesuch copper-mine, at Lake Superior, in which he was successful. But he maintained his general practice as consulting mining engineer in Denver.

His last important problem was the application, in 1908, of the Longmaid-Henderson process to the treatment of Sudbury nickel-copper ores. He showed how, by varying the usual mode of operating, the copper could be rendered soluble while the nickel remained insoluble, and smelting the residual iron oxide for nickel-bearing pig-iron would furnish a raw material for making nickel-steel in the open-hearth furnace.

The same year, 1908, he became a charter member of the Mining and Metallurgical Society of America.

This is, in brief, an outline of the life and career of a man who was always in the front where work was to be done, who by hard work and study prepared himself for the duties that were ahead of him, who had the respect and esteem of all who came into business relations with him, and the personal love of all who knew him intimately.

He leaves a widow, three grown-up sons who have followed the father's profession, one minor son, and a daughter.

Below is appended a list of his papers and a partial list of his patents. The finished papers are not numerous; the short, signed articles which he wrote for current newspapers and periodicals, to correct erroneous reports, or to enlighten popular ignorance, would, if collected, doubtless amount to several hundred in number.

Dr. Carpenter became a member of this Institute in 1887, and his contributions to the *Transactions*, given in the list

below, comprise two papers of wide scope and permanent value.

List of Papers.

	Vol.	Page.	Date.
American Institute of Mining Engineers:			
Ore-Deposits of the Black Hills of Dakota,	XVII.	570	1888–89
(With W. P. Headden.) Note on the Influence of Columbite upon the Tin Assay,	XVII.	633	1888–89
Pyritic Smelting in the Black Hills,	XXX.	764, 1128	1900
American Geologist:			
A New Glacial Theory,	III.	138	1889
Colorado Scientific Society:			
The Separation of Gold from Copper, with Especial Reference to Pyritic Smelting,	VII.	79	1901–04
The New Geology and Vein Formation,	VII.	289	1901–04
Smeltery-Fumes and Their Condensation,	IX.	65	1908–?
Dakota School of Mines:			
Notes on the Geology of the Black Hills, and Mineral Resources of the Black Hills (Preliminary Report),			1888
Engineering and Mining Journal:			
Dakota Tin-Mines,	XLIV.	57	1887
South Dakota,	LIII.	59	1892
The Dry Separation of Gold and Silver,	LXV.	193	1898
Raw Sulphide Smelting,	LXXVI.	688, 844	1903
Negative Results in Pyritic Smelting,	LXXXIV.	939	1907
	LXXXV.	220	1908
Mineral Industry:			
Pyritic Smelting,	IX.	694	1900
Progress in Pyritic Smelting in 1901,	X.	695	1901
Mining and Scientific Press:			
Mine Fires,	XCVI.	697	1908
Mining World:			
Tin in the Black Hills, South Dakota,	XXV.	600	1906
United States Treasury Department:			

Production of Gold and Silver in the U. S. Reports for South Dakota, 1896–1905, inclusive.

List of Patents Granted.

No. 344,720, June 29, 1886: Machine for Concentrating Ores.
No. 499,318, June 13, 1893: Process of Treating Ores.
No. 597,139, January 11, 1898: Art of Separating and Refining Metals.
No. 678,457, July 16, 1901: Process of Treating Ores.
No. 718,087, } January 13, 1903: Process of Separating Precious Metals from Their
No. 718,088, } Ores.
No. 718,089, January 13, 1903: Process of Recovering Precious Metals from Mattes Containing Them.
No. 718,601, January 20, 1903: Process of Separating Precious Metals from Matte.
No. 722,809, March 17, 1903: Method of Treating Ores.
No. 735,032, July 28, 1903: Process of Separating Precious Metals.
No. 781,807, February 7, 1905: Process of Smelting Native-Copper Bearing Rock.
No. 781,808, February 7, 1905: Process of Reducing Vanadium.
No. 829,765, August 28, 1906: Process of Recovering Sulphurous Oxide.
No. 871,912, November 26, 1907: Apparatus for Obtaining Sulphur from Furnace-Gases.
No. 904,838, November 24, 1908: Process of Treating Metalliferous Ores.
No. 922,388, May 18, 1909: Process of Making Ferro-Nickel and Nickel-Steel.
No. 925,751, June 22, 1909: Process of Obtaining Sulphur, etc., from Furnace-Gases.
No. 959,841, May 31, 1910: Process for Rendering Potash Compounds Soluble.

List of Patents Pending.

Serial No. 303,905, filed March 2, 1907: Process of Obtaining Compounds of Aluminum and Potassium.
Serial No. 358,442, filed March 20, 1907: Process of Recovering Sulphurous Oxide from Gases Issuing from Furnaces.
Serial No. 439,212, filed June —, 1908: Process of Concentrating Gases.
Serial No. 500,922, filed June 8, 1909: Manufacture of Ferro-Nickel Alloys.
Serial No. 514,552, filed August 25, 1909: Treatment of Copper-Nickel Ores.
Serial No. 514,992, filed August —, 1909: Improvements in Processes of Making Ferro-Nickel and Nickel-Steel.

DISCUSSIONS.

Ozark Lead- and Zinc-Deposits; Their Genesis, Localization, and Migration.

Continued discussion of the paper of Charles R. Keyes, *Trans.*, xl., 184, 856.

CHARLES R. KEYES, Des Moines, Iowa (communication to the Secretary *):—With regard to the fancied absence from my paper of sufficient reference to the work done by Mr. Buckley from the Bureau of Geology and Mines of Missouri, I beg to say that his principal publication appeared some months after my paper was finished and sent to the Institute (September, 1908) for publication. By reason of the delays incident to the reading of proofs, the preparation of engravings, the intervention of the Chattanooga meeting (at which the paper was nominally presented for subsequent publication), and the demands of other contributions submitted earlier, and therefore entitled to "the right of way," my paper did not actually appear in the *Bulletin* until February, 1909. Yet, since it was published as a paper of the Chattanooga meeting of October, 1908, it cannot be fairly condemned for the omission of references to publications of later date than September of that year.

Moreover, I did not feel myself bound to present in such a summary either an exhaustive discussion of the origin of the Ozark ores, or elaborate details of the proofs of my statements on various points—such proofs having been published elsewhere already, and being fully indicated in the numerous footnotes of my paper. At all events, I wish to disclaim heartily any intention of ignoring Mr. Buckley's work or the publications of the Missouri Bureau of Geology and Mines, to which he has contributed. I quoted this authority as frequently and fully as I thought its work, so far as it was then known to the public, deserved—as often, in fact, as I quoted any other authority. But I did not then, and do not now, accept its con-

* Received Apr. 30, 1910.

clusions as final, or as entitled to greater weight than those of geologists who, unbiased by industrial interests, have studied the Ozark region.

Before replying in detail to Mr. Buckley's criticisms, I wish to point out that, in some of them at least, he has mistakenly attributed to me what I did not say. Thus, on p. 861, referring to Fig. 10 of my paper, he says that I represent the ore as occurring at the base of the La Motte sandstone, whereas, "any one who has ever been in the disseminated-lead district knows that the ore now being mined occurs above the La Motte sandstone, and that only where the sandstone is absent does it rest upon the granite." Without presuming to question the somewhat remarkable assertion that "any one who has ever been in" that district knows where the ore "now" mined comes from, I may be permitted to point out that my Fig. 10 was taken from the reports of the Missouri Geological Survey of 1895; that it does not represent the ore-bodies as lying under the La Motte sandstone; and that I have repeatedly asserted the contrary. (See *Missouri Geological Survey*, vol. ix., pt. iv., p. 48, 1905.)

Again, Mr. Buckley's criticism of my Fig. 8 is not a criticism of what I really represented, but of his own interpretation, without consideration of the specific foot-note reference to Mr. Winslow's well-known Osage River large-scale section of 200 ft. to the inch (*Missouri Geological Survey*, vol. vi., p. 358).

Again, "because this [central district] is a cavernous region it is not to be assumed that it was once a richly-mineralized region" is an assumption of Mr. Buckley's, not any special generalization of mine. Nor is there any statement of mine that I can see should warrant this interpretation of my conclusions by Mr. Buckley.

Again, regarding the lack of proofs that the "genetic relationship of ore-runs to buried relief-features at the base of the Coal Measures does not obtain" there might have been further quotation in order to give the completed meaning. However, the best demonstration of the correctness of my statement is to be found in Mr. Buckley's own report of the Granby Area.

Again, the proof of the non-existence of major faulting in the Joplin district is much older than the appearance of Dr. Bain's memoir. Mr. Winslow (*Missouri Geological Survey*, vol.

vii., p. 430) is quite specific on this point; and his work antedates the report on the Granby Area by a decade.

Further, Mr. Buckley's statement that I "evidently conclude, as others have done, that because faulting is not associated with the ore-deposits of the Joplin district, it is safe to conclude that this structure has no relation to the ore-deposits of other districts in this region" seems gratuitous, in view of the fact that in my paper I devote a considerable part of it to urging the importance of faults.

To Mr. Buckley's repeated demand for the evidence upon which my statements were founded, I would make the general reply that, apart from my own personal observations, the reports of the *Missouri Geological Survey*, vols. i. to xii. (1891 to 1897), contain the information which he desires. Full references are given in foot-notes in my paper; but I am glad to assist him with the specific references contained in the following paragraphs, each of which begins with the citation of the page of my paper in the *Transactions.*

P. 192.—The full reference to the Ozark dome and its characteristics is the *Missouri Geological Survey*, vol. viii., pp. 317–352, where I have fully and especially described the structural aspects of the region, with particular reference to their bearing upon the ore-deposits.

P. 203.—The proof of the relation of the Joplin ore-bodies to basins and not to free underground channels is amply attested by the detailed contour-maps of the district, which were prepared by the Missouri Geological Survey under Mr. Winslow, and by the later similar contour-maps incorporating the work of Messrs. Smith and Siebenthal, and published by the United States Geological Survey as Folio 148.

P. 196.—"First concentration" is explained on the next page of my paper by special reference to the work of Van Hise and Bain (*Transactions of the Institution of Mining Engineers*, vol. xxiii., p. 401).

P. 199.—Upon reference to the general map of the lead- and zinc-areas of Missouri (*Missouri Geological Survey*, vol. vii., p. 700), it will be noted that the central district lies entirely on the north slope of the Ozark dome and nearer its margin than its crest.

P. 195.—The location of "all the known mines [in the cen-

tral Missouri district] in certain straight and narrow belts, which are near and parallel to the axes of shallow synclines . . . pitching radially from the center of the uplift," is clearly shown in the maps of the central district (*Missouri Geological Survey*, vol. vii., p. 700), taken in conjunction with the Osage River geologic section (*Ib.*, vol. vi., p. 358). My own more recent investigations in this district fully substantiate my earlier observations. I am not at liberty at this time to publish my own later maps and the detailed data elucidating the facts.

P. 230.—The retreat of the margin of the Coal Measures down the slopes of the Ozark dome in no way militates against the possibility of the existence of extensive ore-bodies at the deeper levels in the central district. Even a casual comparison of levels shows the two statements to be mutually corroborative (see *Missouri Geological Survey*, vol. x., p. 24). The location of the Joplin syncline is indicated on most of the geological maps of Missouri. It is especially emphasized by Mr. Winslow (*Missouri Geological Survey*, vol. vii., p. 485), in the discussion of his theory of ore-concentration. The various basins and synclines about Joplin are clearly shown on the Smith-Siebenthal map (*Geologic Atlas of the United States*, folio 148). The Joplin syncline is most carefully represented on Mr. Buckley's map, which he copied directly from mine.

Recent investigations support my position regarding the southeastern district more strongly than even my statement does. Mr. Buckley's late memoir furnishes more evidence against his own theory than in support of it. All of the practical tests made during the past 15 years in this district fully confirm my original working hypothesis, as the subsequent developments in Mr. Buckley's work amply attest.

I think that the fundamental differences which Mr. Buckley entertains are unmistakably disclosed in the third paragraph of his criticism. In substance, his deduction is that the ore-bodies are localized in trunk-channels, where the free and copious circulation of dilute metallic solutions comes into contact with reducing agents. This is an inference as plausible as it is simple; but it contains only a half-truth. Its necessary consequences are at utter variance with recorded facts. It is a result of backward reasoning from accidental effects to sup-

posed cause. It ignores completely the broader geologic relationships, conditions, and processes.

In its bearing upon ore-localization, the most important effect of the elevation of a broad area of horizontal and soluble strata is its subjection to rapid reduction towards base-level (and the Ozark uplift undoubtedly presents these conditions at the present time), and the development of a free and copious radial flow of the drainage-waters through an exceptionally open vadose zone, as is amply attested even in the characteristic Karst topography. In the Ozarks ore-deposition of to-day must necessarily be in the main relatively unimportant and largely near the base of the dome or in the locally protected places where there is temporary impounding of the ground-waters. With this deduction the facts seem strictly to accord. The ores of the Ozarks appear to be, at the present time, rapidly dissolving and passing away instead of being abundantly deposited. This statement seems to find ample support in the results of chemical analysis of the phreatic waters of the region. The ore-deposits as they are seen to-day are, then, in the last stages of complete disappearance. The open brecciated ground near Joplin, Webb City, and Oronogo furnishes the most indisputable testimony.

As to the sources of my information, I pass over for the present further reference to my own geologic work in the Ozark region; but it is pertinent in this connection for me to make comparison of the painstaking and strictly scientific reports of Mr. Winslow's with the critic's own prolix discussions.

I am free to say that Mr. Winslow's lead and zinc reports (Vols. VI. and VII. of the *Missouri Geological Survey*, 1894) are models of what geologic reports on mineral deposits should be. There is nothing on Ozark ore-deposits that at all compares with them. These reports are in the highest degree scientific. Every page breathes of engineering acumen. The subject-matter is thoroughly digested, tersely presented, and happily illustrated. In the consideration of moot questions all evidence is carefully and clearly set forth for the reader himself to judge and to draw his own conclusions. Vast stores of information and observations are condensed into small and presentable form. For plain, unvarnished facts regarding the Ozark lead- and zinc-deposits

one must always turn to the Winslow reports. They are to-day the best sources of exact information that we have. A generation hence they will be as fresh and as helpful as the day on which they left the press. Nowhere in them do we have to labor long and patiently, and often in vain, in order to disentangle fact from fancy. One closes the books with his mind refreshed, and with a feeling that much has been gained by perusing them.

Throughout the Winslow lead- and zinc-reports there is a complete absence of the controversial spirit. It is possible to form from the facts recorded one's own mental pictures. A hundred years from now, one will be able to glean from them critical facts concerning the Ozark ores, although view-points shall have entirely changed and new theories shall have been accepted. These reports are not statements for the day merely, to be rewritten and republished to-morrow; they are books for the decade and the century. From the stand-point of the miner, the metallurgist, the engineer, and the scientist, nothing that has appeared on the subject since they were issued comes up to them.

Not the least valuable feature of the Winslow reports on the lead- and zinc-deposits of Missouri is the clear recognition of the fact that hypotheses of ore-deposition are only valuable in proportion as they can predict results and point out new ore-fields. Than theory in this sense, nothing is more completely practical. It is a phase of report-building that all workers can follow with great advantage. For this reason, if for no other, these reports are of unusual permanent value. For this reason it is that their intrinsic worth is so exceptional. For this reason, also, it is that their high merit cannot be suppressed by simply ignoring them. Mere citation of them in the discussion of the Ozark ores is but adding luster to one's work.

A crucial test of any hypothesis in geology is measured quantitatively by its value in prognostication. Mr. Buckley to the contrary notwithstanding, my own generalizations have rigidly stood every practical test in the field, and in no case that I know of have those which the critic has set forth with the seal of approval of the Missouri Bureau of Geology and Mines. Of this the future may also judge. So far as concerns the Ozark ore-deposits at least, Mr. Buckley's case is utterly hope-

less until he shall have trained himself to meet other workers in the same field in a spirit of scientific and professional tolerance, and shall present his own evidences of observation with something of that judicial impartiality that so strongly characterizes those monumental reports of Arthur Winslow.

The great and fundamental difference between Mr. Buckley's ideas of the Ozark deposits and my own "theories" is, then, that when practically tested in the field they have in the one case utterly failed and in the other they have not.

Metal-Losses in Copper-Slags.

Discussion of the paper of Lewis T. Wright, *Trans.*, xl., 492.

J. PARKE CHANNING, New York, N. Y. (communication to the Secretary*):—Mr. Wright, in his introductory paragraph, says:

> "It is commonly believed by metallurgists that in copper-smelting, the copper in the slags, which is irreducible by continued smelting, is retained in the form of 'prills' of matte."

On a recent visit to Greenwood, B. C., I was discussing with J. E. McAllister, the general manager of the British Columbia Copper Co., and the former metallurgist of the Tennessee Copper Co., this very question; and he held strongly to the idea that the copper in slag was in two forms—one portion contained in occluded matte and the other as an oxide. He believed that this same rule would apply to the silver in the slag.

From my experience I am strongly inclined to agree with Mr. McAllister; and it appears quite reasonable that in furnaces treating oxidized copper-ores, and to a less degree in furnaces treating sulphide ores, there is always bound to be present a certain amount of copper oxide and silver oxide, which will behave like any base and get into the slag as such.

We all know that in Arizona, in the early days, when it was customary to produce black copper, because the ores were nearly all oxidized or carbonates, and sulphur was scarce, the slags as a rule carried 2.5 per cent. of copper and seldom ran

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*Received Oct. 8, 1909.

less than 1.5 per cent. of copper. The economic point was determined by the extra coke necessary to produce the reducing action. In connection with the extra saving in copper, Arthur L. Walker, now professor of metallurgy at Columbia University, and formerly general manager of the Old Dominion Copper Mining Co., at Globe, told me that with coke at $60 per ton it never paid him to make slags containing less than 2.5 per cent. of copper.

At Copperhill, Tenn., in treating well-roasted and presumably well-oxidized ore, we seldom, if ever, had the slags run less than 0.5 per cent. of copper, yet in treating this same ore pyritically and producing the same grade of matte, the slag would not exceed 0.3 per cent. of copper.

The above instances are stated merely to show that the normal tendency of copper oxide is to act like iron oxide, lime, or any other base, and go into the slag.

If we take Mr. Wright's first example and assume a 50 per cent. copper-matte corresponding with 0.3 per cent. copper-slag, which, by the way, would be pretty low, we would have the following analyses of the matte and slag:

	Matte.	Slag.
Cu, ...	50.000 per cent.	0.300 per cent.
Ag, ...	31.400 oz. per ton.	0.147 oz. per ton.
Au, ...	13.950 oz. per ton.	0.026 oz. per ton.

If we are permitted to assume the hypothesis that no gold is oxidized in the slag, and that all there present is contained in the matte, we can assume the 0.026 oz. per ton as a basis and from it calculate the amount of silver and copper corresponding to the grade of the matte produced. I have given below, in the second column, the results of this calculation:

	Total Loss.	Matte Loss.	Oxide Loss.
Cu, .	0.300 per cent.	0.093 per cent.	0.207 per cent.
Ag, .	0.147 oz. per ton.	0.058 oz. per ton.	0.089 oz. per ton.
Au, .	0.026 oz. per ton.	0.026 oz. per ton.	0.000 oz. per ton.

This calculation shows that of the total copper contained in the slag about one-third only is lost in the form of matte and two-thirds is probably in the form of oxide. Very nearly the same ratio exists for the silver; and we all know that silver is easily oxidized in a furnace with a hot top, such as is usual in

copper-smelting, and, while no doubt part of it is volatilized, a portion of it is ready to go into the slag.

While neither Mr. McAllister nor I has any means of actually verifying the above theory, it appears to us the simplest and most obvious solution of the problem, and much more in keeping with the facts than the suggestion that the metals themselves are dissolved in the reject material from the furnaces, which is a mixture of slag, containing a very small proportion of matte.

The Genesis of the Leadville Ore-Deposits.

Discussion of the paper of Max Boehmer, p. 162.

W. Morton Webb, Germiston, Transvaal, South Africa (communication to the Secretary*):—The experience of Mr. Boehmer in the Leadville district and his reputation as an engineer assure the interest of anything he may write on the ore-bodies of that district; and all engineers who have operated in Leadville will agree that the theory of the origin of the Leadville deposits, as given by Mr. Emmons in his famous monograph, does not adequately explain the genesis of all the deposits. The solution of the problem does not, however, appear to be quite so simple as Mr. Boehmer would make it; and the advance pamphlet issued by the U. S. Geological Survey, pending the completion of the main report, shows that there is still much ground for controversy.

No doubt the later eruptives had a greater influence on the deposition of the ore-bodies than was thought at the time of the first investigation by the Survey; but it has always appeared to me that the influence exerted by these dikes was more in the nature of a physical than a chemical one. The weight of evidence in recent years is mostly in the direction of ascension rather than lateral secretion; and one of the strongest evidences is the fact that most of the ore-bodies lie under one of the comparatively impervious layers which help to make up the structure of these hills.

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* Received Mar. 3, 1910.

On Iron hill I have found a very persistent white porphyry sheet, split off from the overlying porphyry and lying from 30 to 60 ft. below the main white porphyry and blue limestone contact. This sheet varies from a few to 50 ft. in thickness, and where found adds another "contact" to that region; some of the largest ore-bodies on Iron hill lie immediately under this sheet. Above this is the main white porphyry and blue limestone contact, while below it again are the "contacts" under a gray porphyry sheet in the blue limestone, under the parting quartzite, under a gray porphyry sheet in the white limestone, and even in the more soluble strata of the transition-beds between the Cambrian quartzite and the white limestone. That all these successive ore-bodies on the various contacts lie usually vertically one below the other, indicates that a line of weakness probably allowed the mineral-bearing solutions to rise and, at the various contacts under the comparatively impervious beds or sheets, to spread out along the planes of least resistance.

That the gray porphyry dikes follow more or less closely the direction of the ore-shoots, as first shown by A. A. Blow in his valuable paper,[1] would point to the influence of the dikes on the position of the ore-bodies; the dikes, by forcing their way through the sedimentary beds, having produced shattered zones more or less parallel with their direction, which later allowed the passage of the mineral-bearing solutions.

Mr. Boehmer says that the work of the past five years on Breece hill has disclosed fissures which solve the question of the genesis of the Leadville deposits. As early as 1900, I studied and mapped these fissures and followed them down continuously into the Cambrian quartzite; but up to the time of my departure from Leadville in March, 1908, none of them had, to my knowledge, been found lower than the lower quartzite. The evidence, therefore, up to that time, while presumptive, was not conclusive; and I have observed evidences that some of the so-called fissures were more likely gashes or cooling-cracks in the eruptive rocks than pre-mineral fault-planes. Some of the main cracks, however, are without doubt pre-mineral faults, and flat

[1] Geology and Ore-Deposits of Iron Hill, Leadville, Colo., *Trans.*, xviii., 145 to 181 (1889-90).

channels of ore have been thrown off from them in the transition-beds at the upper part of the Cambrian quartzite, in the white limestone, in the blue limestone, and even in the micaceous quartzite-beds of the Weber Grits series. This last horizon was worked by the Ibex Mining Co. from 1901 to 1903, and was a very valuable deposit, extending several hundred feet horizontally into the hanging-wall of the parent fissure. The fissure here, as well as the flat channel, contained a large quantity of secondary chalcocite; and, while the fissure had a maximum width of perhaps 4 ft., the flat deposit at 200 ft. from the fissure was more than 10 ft. thick. Other "flats" also were thrown off from both the hanging-wall and the foot-wall of this fissure; and the phenomenally rich gold-ore found in 1907 by one of the lessees near Ibex No. 4 shaft, was from one of the flat seams in the transition-beds.

Mr. Boehmer says that the flat mineralization extends at least a mile from the fissures, and from this concludes that the deposits in the western part of the district had their origin in these fissures on Breece hill. While he may have satisfactory evidence of this, I have never been able to trace any flat body directly for more than a few hundred feet from a fissure; and I am inclined to think it is asking a little too much to place to the credit of these comparatively small fissures on Breece hill the immense deposits lying under the city of Leadville itself and extending three or four miles from their accredited source.

The Combustion of Coal.

Discussion of the paper of Joseph A. Holmes and Henry Kreisinger, p. 244.

WILLIAM KENT, Montclair, N. J. (communication to the Secretary*):—The authors say, "The factor which determines the completeness of combustion of the volatile matter, after it has been mixed with a certain amount of air, is the length of time the mixture is allowed to remain in the combustion-space; but this length of time depends on the extent of the space itself."

* Received May 31, 1910.

This may be true under a given set of conditions, such as those existing in a Murphy stoker furnace, operated with natural draft, and with an imperfect mixture of gases and air. Under other conditions, however, such as those of a complete and homogeneous mixture, the time-factor and the space-factor may be reduced to almost nothing. For instance, a mixture of hydrocarbon gas and air in the proper proportions may be fired by an electric spark with practically instantaneous and complete combustion, in fact an explosion.

The chief factor which determines the length of the flame of coal-gas is the completeness with which the air supplied for combustion is mixed with the gas. I have seen a flame 12 in. long proceeding from a Bunsen burner, when the air-holes at the bottom were closed, and the gas received air for combustion only at the surface of the flame. On admission of air at the bottom the flame was shortened more than half, and its color changed from reddish-yellow to blue. On mixing the air and gas in a small centrifugal fan before admitting them to the burner, the flame was less than 2 in. long, and the tip of it was hot enough to melt a platinum wire.

Long combustion-furnaces may be needed for the complete burning of soft coal when the gases and air are imperfectly mixed, but they are not necessary if the furnace makes proper provision for securing a thorough mixture. I once tested a Root boiler, burning Ohio coal containing more than 40 per cent. of volatile matter. The boiler was driven above its capacity, but the flame was so short that it did not reach the tubes, although they were directly over the fire, and there was no smoke from the chimney. The stoker was the old style American Underfeed, with a forced draft. I have seen a Babcock & Wilcox boiler driven at double its rating, using a semi-bituminous coal containing 20 per cent. of volatile matter, and without smoke, the flame being transparent a few inches above the fire. The Taylor Gravity Underfeed stoker was used. On the other hand, I have seen a Heine boiler tested, using semi-bituminous coal, when the flame was so long that the combustion was not complete until the flue-connection was reached, which was made red-hot by the burning gas. This also was a test of an underfeed stoker, but the inventor of it had not put the air-holes in the proper place, so that the air did not mix

with the gases in the furnace, but traveled parallel to them, getting partly mixed as they passed along between the tubes.

The length of the flame in a boiler-furnace is a measure of the imperfection of the furnace rather than of the quality of the coal. The proposed government tests to determine the length of flame proceeding from different kinds of coal will be of little value unless they are supplemented by other tests to show how the flame from any kind of coal may be shortened by making provision for the complete mixture of the gases and air before they enter the combustion-space. The Murphy furnace will probably have to be replaced by one of the underfeed stokers before this can be done satisfactorily at rapid rates of driving.

When the mixture of gas and air is imperfect, another factor upon which the length of flame depends is the temperature of the combustion-chamber; the hotter the chamber the shorter the flame. Still another factor is the percentage of moisture in the coal. The evaporation of this moisture tends to chill the fire, and in ordinary furnaces the production of water-gas from this moisture tends to retard the mixing of the gas and air.

A Commercial Fuel-Briquette Plant.

Discussion of the paper of W. H. Blauvelt, p. 255.

Charles T. Malcolmson, Chicago, Ill.:—Mr. Blauvelt's admirable paper is a valuable contribution to the literature on briquetting of coal in this country. It should have a special significance for those interested in the commercial development of the coal-briquette industry. I believe that the data included in Mr. Blauvelt's paper will be particularly useful because they represent costs taken from a briquette-plant in actual operation.

There are several points touched on in Mr. Blauvelt's paper which may bear comment in the way of emphasis rather than of criticism. In the first place, he has stated a fact which becomes axiomatic to any one who has had any experience in the design, building, or operation of coal-briquette plants, and that is, that the press is only one factor in the successful opera-

tion of such a plant. I have observed from my experience and from my knowledge of the troubles of other briquette-plants that practically all the failures can be attributed to a disregard of this fact. The successful plant is the one in which all of the elements making up the plant have been given due consideration. The preparation of the coal before briquetting is of more importance in an economic way than the press used in making the briquettes, except in so far as the type of press determines the size or shape of the briquettes made.

Coal-briquetting is a simple operation if viewed from the steps in the process. Coal and a suitable binder are mixed together, heated, and subjected to pressure. The exactness with which these relations are maintained determines the uniformity of the product, and has a vital bearing on the cost of making this product. In the first place, the coal must be dry, and the binder must be introduced into the coal with as little moisture as possible. Experience has shown that a certain percentage of moisture is necessary to make good briquettes, but if the moisture-content exceeds 5 per cent. the product deteriorates in value. The percentage of moisture in the mixture varies with different coals; therefore, that part of the plant is designed to meet the requirements. The temperature of the mixture is also varied with different fuels. These items have a direct bearing on the cost, so that the temperature is kept as low as is compatible with good operation. Mr. Blauvelt says that about 200 lb. of steam per ton of briquettes is required for heating the mixture. At the plant of the Standard Briquette Fuel Co., at Kansas City, we are preparing to install a gas-meter recently designed by Professor Thomas for measuring the quantity of superheated steam used in this plant for this purpose. Unfortunately, at the present time, we have not secured this data, but from other data obtained I believe the figure given by Mr. Blauvelt is high for the Kansas City plant. The quantity of steam, however, will vary according to its temperature. Some operators in Europe disregard the use of superheated steam other than that necessary to insure perfectly dry steam at the mixer.

On the other hand, there are plants at which the coal is heated before the binder is introduced, and the temperature of the mixture is also increased by direct heat, together with the

introduction of steam into the mixture. There are objections to this method in the practical operation of a plant on account of the fire-risk and the caking of the mixture on the inside of the heaters. In the Kansas City plant we use the vertical type of heater mentioned by Mr. Blauvelt, and believe it has some points in its favor. It insures a more or less compact mass, in which the maximum temperature of the heating-agent is applied near the point of discharge of the mass from the heater, while the mixture as it enters the heater at the top receives the excess or uncondensed steam at atmospheric pressure after it has given up its superheat. It also allows the introduction of the steam into the mass under pressure, which increases the total heat in the steam.

The most important problem which we have yet to solve in connection with commercial briquette-plants is the binder. This is by far the largest single item of cost in the briquetting of coal, and the success or the failure of a plant may depend, first, upon the cost of binder, and second, upon its uniform quality. In Europe the briquetting of coal has been developed through many years along with other means of utilizing fine or waste coal. This is notably true of the by-product coke-ovens. There the briquette industry consumes a large portion of the tar recovered in these ovens. As the consumption of this tar represents an important item in the commercial success of this form of coke-ovens, and the briquette industry consumes the major portion of the tar so produced, the requirements for briquetting-pitch have been met by the producers of by-product coke. The two industries must of necessity work hand in hand. I cite the coke-oven industry because this is the largest producer of tar, the pitch of which is used mainly for briquetting purposes.

In the United States the tar, whether it be recovered from by-product coke-ovens, from the manufacture of illuminating-gas by destructive distillation of coal, or from tar recovered in the washing of water-gas enriched with petroleum, has found markets in other channels, notably in the roofing and paving industries. These industries have not required the exactness in the manufacture of pitch which is essential to the making of good briquettes. For instance, it is a common practice in the United States to "cut back" a still of pitch if the distil-

lation has been carried beyond the proper point to produce pitch of a given melting-point, or hardness. This means that oils are introduced into the still to soften the pitch after the fires are drawn.

We have learned by experience a fact which has been known in Europe for some time—that there are certain oils in coal-tar which must be removed before the pitch can be used for the manufacture of briquettes, particularly for domestic purposes. If these oils are retained in the pitch, they volatilize at a temperature below that at which the coal burns, and in consequence smoke forms, which is not only objectionable on account of the odor, but the oils contained in it often condense in the cooler chimney, collecting the finely-divided carbon of the smoke and obstructing the draft. When briquettes made with pitch of this character are used on locomotives, the dust and fumes from the briquettes irritate the faces and eyes of the firemen.

In order to develop the industry to a point where the proper briquetting-pitch can be specified and maintained at all times, considerable investigation will be necessary, the cost of which must be reckoned in the cost of the finished product. At the present time the low selling-price of fuels in the United States leaves small margin for an excessive cost of binder, since briquettes must be made at a figure between the selling-prices of the slack- and lump-coal and leave a margin of profit.

These problems have yet to be worked out commercially; but there is much encouragement from the attitude of the large tar-distillers in the United States, who have given every evidence of their willingness to bear their share of the burden.

There is a point only touched on by Mr. Blauvelt in his paper, regarding the advantage in the use of hard or dry pitch, as compared with a softer pitch introduced into the mixture in liquid form. Experience has shown that a pitch of higher melting-point does not work so well with the liquid process. A proposition, however, of larger economic value for the future is found in using a hard pitch which can be shipped in bulk in all seasons and which will give the tar-distiller the more valuable oils resulting from the manufacture of such pitch. In Europe the practice is quite general of distilling the pitch beyond a point at which it can be used even for briquetting pur-

poses in order to obtain the valuable anthracene oils. This hard pitch is "revivified" by the admixture of a softer pitch. The hard pitch is always shipped in bulk; the softer pitch must be handled in barrels, which adds to its cost. In the United States the manufacture of aniline dyes, alizarine, and other chemicals obtained from the anthracene, has never become an important factor. Most of these chemicals are imported from Europe.

The development of this industry will undoubtedly follow the introduction of practical methods for the effective conservation of our fuel-resources, in which briquetting will play an important part.

JOHN BIRKINBINE, Philadelphia, Pa.:—Mr. Blauvelt's general reference to the numerous forms of briquette-apparatus brought to public notice in this country, as recorded in our *Transactions* and other technical publications, might have been elaborated and made exceedingly instructive by detailed mention of the different forms and the business enterprises connected with their introduction, and a classification of them into those which were mechanically defective; those which were mechanically successful, but not economical; and those which were simply premature, because of the lack of public appreciation of their importance as methods of utilizing waste material.

With regard to the history of this question, as exhibited in our *Transactions*, I may supplement Mr. Blauvelt's statement by calling attention to the fact that, at the Philadelphia meeting of the Institute, in February, 1872, the late E. F. Loiseau presented orally a paper on the utilization of coal-dust,[1] which, however, was not published, Mr. Loiseau preferring to present later and more complete accounts[2] of his briquetting-method and his plant at Port Richmond, near Philadelphia. Meanwhile, in connection with the Hazleton meeting of October, 1874, the members of the Institute visited, near Mauch Chunk, the experimental works of Mr. Loiseau, who afterwards erected the larger plant at Port Richmond.

[1] *Trans.*, i., 18 (1871–73).

[2] On the Manufacture of Artificial Fuel at Port Richmond, Philadelphia meeting, February, 1878, *Trans.*, vi., 214 (1877–78), and The Successful Manufacture of Pressed Fuel at Port Richmond, New York meeting, February, 1880, *Trans.*, viii., 314 (1879–80).

In addition to these accounts of Loiseau's process and machinery, our *Transactions* contain, besides references to the briquetting of ores, etc., an elaborate paper by Robert Schorr, of San Francisco, Cal.,[3] on Fuel- and Mineral-Briquetting, the value of which is enhanced by a comprehensive bibliography, prepared by the author, and complemented with important additions by Professor Hofman.

But, apart from the foregoing suggestions for the assistance of those who would study the literature of this subject, I wish to recognize the significance of the data presented by Mr. Blauvelt, as constituting a welcome contribution to "the conservation of natural resources," by proposing a method of producing cheaply a satisfactory fuel from what is, in many localities, a refuse material; and I desire also to point out a special field in which such a method may be greatly useful.

The anthracite coal-fields of Pennsylvania have long been looked upon as offering inducements for some method of utilizing the fine coal. In late years many washeries have been installed to recover small sizes of anthracite from mine-waste, and numerous old culm-banks have been worked over for this purpose. But these washeries do not reach great quantities of coal which have been washed down the streams of the coal-region into the large creeks and rivers; and, in these, a considerable industry is carried on by flat-boats equipped with dredges and screens, which recover large amounts of fuel from stream-beds. The dredges win good coal from "pea"-size up, and the washeries save what is marketed as "rice"-coal (names indicating the approximate sizes); but the coal-dust is not recovered in either case, and the operation of the machines largely augments the amount of this which enters the streams.

The Water Supply Commission of Pennsylvania has received protests against the discharge into the streams of this fine anthracite and slate; and, as a member of that Commission, I welcome any practical suggestion for utilizing material which is obstructing stream-beds, doing great injury to cultivated lands, and polluting public water-supplies. This fine coal is admirably suited for the manufacture of briquettes, because the coal grinds to a powder more easily than the slate, and the

[3] *Trans.*, xxv., 82 (1895).

difference in specific gravity permits a fair separation by simple sedimentation. At present, the "sludge" leaving the washeries is carried down the streams where the velocity is sufficient to transport it, but settles in quiet pools, obstructs the stream-flow, and changes the course of the channels. Freshets, lifting the mass, carry it forward and deposit it on cultivated fields, effectually killing their fertility. With each succeeding freshet the obstructed stream-channels force the freshet-waters, laden with coal-dust, over new areas of bottom-land, and, like a moraine, this wave advances down-stream to ruin new territory.

The conditions described in the anthracite-region are to a greater or less extent repeated in the bituminous-coal districts of Pennsylvania, and the condition of many streams of the State is deplorable. Our Commission faces the problem of reducing the permanent injury to some of the best bottom-lands, the obstruction of streams, and the pollution of water-supplies, without working unnecessary hardship to the magnificent coal industry of the State. If the fine coal can be removed from the sludge and cheaply formed into briquettes, this measure may be a great public service. Perfect separation of finely-divided coal from slate-dust may not be necessary, for while the percentage of ash-producing material will influence the calorific and commercial values of briquettes, they may contain a considerable proportion of the slate and yet be a desirable fuel. The problem is not easy. There are difficulties in collecting the fine coal, and the expense of briquetting must be considered; but the field suggested above is worthy of investigation by those interested in briquetting, since the raw material is now not only a waste, but also a nuisance, and a serious menace.

R. W. Raymond, New York, N. Y.:—Speaking from my general recollection of the enterprise of Mr. Loiseau, to which Mr. Birkinbine has referred, and without pretending to know its commercial history, down to the death of the inventor in 1886, I may say that the opinion which I formed of it at the time would fairly place it among those otherwise excellent methods which Mr. Birkinbine characterizes as prematurely proposed. There was no question as to the quality of the

product; but the process required one condition, upon which many other, and less meritorious, schemes of the kind were wrecked—namely, it called for pure material. Innumerable philanthropic and visionary reformers had gazed upon the big, black "culm-heaps" of the anthracite-region, estimated them as measuring the "waste of coal," and dreamed of ways in which they could be "utilized." And their various schemes were usually based upon the notion that these black piles of waste would furnish a raw material available for use at once, without other expense than that of handling it. In fact, probably none of these old dumps contained enough coal to be used, without previous concentration, in the form of briquettes or in any other form; the oldest of them, accumulated when small sizes of coal were not marketable, would pay for re-washing; and the great majority, accumulated after the introduction of the use of the smaller, smaller and smallest sizes of coal, were practically too poor in carbon to be economically handled at all. Reformers and inventors alike seem to have regarded every dump of black slate as so much waste of coal.

Two men, more than any others, introduced effective remedies for the waste of coal in this way—namely, Franklin B. Gowen, who, as President of the Philadelphia & Reading Railroad Co., put upon the road of that company locomotives provided with special fire-boxes and grates for the burning of unsalable slack-coal; and Eckley B. Coxe, who was a leader in promoting the use of smaller sizes of coal. Mr. Gowan's innovation had but a moderate success. It is my impression that the Reading Railroad Co. is not now building locomotives of his type; and I am inclined to think that the economy which he sought to secure is now attained in other ways, less inconvenient to railway-service—among which, I fancy, Mr. Coxe's more profound and permanent improvements play an important part.

According to my recollection, Mr. Loiseau was subject to no delusions regarding the culm-heaps as a source of his raw material. He knew very well that he must have practically pure coal for his briquettes; and it was for this reason that he put his plant at Port Richmond, where the screenage, before further shipment, of coal received from the mines would furnish the quality of coal-dust which he required. In this view he

was clearly right; for this secondary screening of coal, already more or less freed from slate in the breaker, certainly gives a waste product immensely richer in carbon than any old culm-heap in the country. There is one other way of obtaining such a product—namely, from the yards of the retail coal-dealers, who, receiving their anthracite by the boat-load, are obliged to screen it once more before delivering it to their customers. And this last screening probably furnishes the purest coal-dust of all. Unfortunately, the amount of such dust accumulated in a single dealer's yard is not likely to be great enough to warrant the erection of a briquette-plant for its utilization; and the cost of handling it and shipping it to a briquette-plant at some other point would be, very likely, enough to prohibit such a disposition of it. In spite of much loose denunciation of the "grasping coal-barons" of the anthracite-region, I venture to say that the price at which anthracite is sold on railroad-cars at the mine, and the price at which it is hauled to tide-water, are astonishingly low, especially when compared with the cost of every subsequent handling of it; and that the utilization of any coal-dust, however pure, which involves its re-handling and re-transportation, at least in a city, makes it, not a raw material, available without cost as "waste," but a material more expensive than coal from the mine.

Without pursuing this argument further, I wish to say that Mr. Blauvelt's paper evidently describes the use of a relatively pure material. It presents no scheme for the preparation of black slate as a fuel. And Mr. Birkinbine's suggestion has in its favor the strong feature, that it indicates, as a source of supply, stream-deposits of a naturally concentrated, and therefore presumably pure, raw material. But an engineer, in considering this suggestion, is impelled to inquire whether any such stream-deposit in Pennsylvania is large enough to warrant the erection of a briquette-plant on the spot, or the establishment of mechanical means for its exploitation and a railroad for its transportation. If this is not the case, then it is more than doubtful whether the material, even if offered free of royalty by the owner of the land, could be dug up, carried away, and delivered to a briquette-plant, at a cost permitting competition with coal from the mine.

Edward W. Parker, Washington, D. C. (communication to the Secretary *):—It may be of interest in connection with Mr. Blauvelt's paper to refer somewhat to the antiquity of the briquette industry. It is being considered as a comparatively new industry, whereas it is really more than 300 years old. I have recently seen a pamphlet printed in Great Britain in 1603, advertising "coal balls" for domestic use. The inventor had in mind two benefits to be secured: one was the utilization of the fine coal otherwise wasted, and the other was the provision of an occupation for persons who had lost their legs or were otherwise incapacitated for performing ordinary labor. The "coal balls" were cemented with tar and molded by hand. The pamphlet describing this invention and the claims of the inventor is in the possession of Prof. J. A. Holmes, of the Technologic Branch of the U. S. Geological Survey.

As is now well known, the history of briquette-making in the United States up to within quite recent years is one of many failures, and it is probable that the amount invested at the present time in practical briquetting-operations does not represent 10 per cent. of the amount invested in hopeless enterprises. The reason for these numerous failures, as I have already stated in a paper presented to the Institute,[4] has been the attempted exploitation of patented and secret binders and processes. To these should be added the ignorance of the mechanical problems which had to be overcome in the development of methods capable of producing a fuel which was not only useful, but which would compete with the plentiful and cheap raw fuel. Fortunately, it has now become generally well recognized that the binder or cementing material to be used is one in which coal-tar forms the principal ingredient and which must be cheap. To my mind, one of the most important and encouraging incidents in connection with the development of the fuel-briquette industry in the United States is the establishment by the well-known engineering firm, Roberts & Schaefer Co., Chicago, of a separate branch devoted to the construction of briquette-plants, of which C. T. Malcolmson is now in charge. This specialization means that it is not now necessary for parties contemplating the building of a briquette-

* Received Mar. 11, 1910.

[4] *Trans.*, xxxviii., 582 (1908).

plant to go ahead blindly in the construction of the plant, which might result in installing mechanical arrangements which are ill-fitted to the work to be performed.

I am looking forward hopefully to the important part that the briquette industry will play in that interesting drama—the conservation of our natural resources.

ROBERT SCHORR, San Francisco, Cal. (communication to the Secretary*):—Mr. Blauvelt says: "Tests at the Detroit plant, extending over a number of days, show a consumption of 206 lb. of steam per ton of briquettes produced. This result was during a period of somewhat slow operation." Mr. Blauvelt would have found the steam-consumption under normal and steady conditions considerably smaller, as he is heating the coke before mixing. In European briquetting-plants, where the hot coal from the drier meets the pitch in the vertical mixer (*malaxeur*), to be subjected to the action of superheated steam, from 28,000 to 35,000 cal. have to be expended per metric ton of briquettes, or from 108,000 to 126,000 B.t.u. per short ton of 2,000 lb. Assuming a coal of 12,600 B.t.u., this would equal from 8 to 10 lb. of boiler-fuel for mixer-steam per ton of briquettes,[1] or equal to from 0.4 to 0.5 per cent.

To arrive at the number of pounds of steam necessary for this purpose, it will be safe to assume an efficiency of 72 per cent. for the boiler and superheater, and 97 per cent. for the pipe-lines, which makes a combined efficiency of about 70 per cent. If we express the heat-requirement in pounds of steam of 100-lb. pressure and of 500° F. temperature (a superheat of 163° F.), which contains a total of 1,283 B.t.u. per pound, we have from $\frac{108{,}000 \times 0.70}{1{,}283} = 60$, to $\frac{126{,}000 \times 0.70}{1{,}283} = 70$ lb. of steam for the mixer per ton of briquettes made. Where the conditions are known it is very simple to figure the amount of fuel or steam necessary for briquetting.

The specific heat of coal ranges from 0.20 to 0.24, and the specific heat of coal-tar pitch from 0.35 to 0.45. It will serve the purpose to consider 0.22 and 0.40, respectively, as averages. In using hard pitch as a binder, it is the practice to heat the

* Received June 28, 1910.

[1] *Trans.*, xxxv., 94 (1905).

mixture to from 80° C. (176° F.) to 95° C. (203° F.). In Mr. Blauvelt's case we have the first-named temperature, and if we assume 92.5 per cent. of coal and 7.5 per cent. of pitch at 56° F. we require theoretically $1,850 \times 0.22 \times 120 + 150$ lb. $\times$ $0.4 \times 120 = 56,040$ B.t.u. per short ton of briquettes. If there were no losses this would mean 44 lb. of steam at 100 lb. and 500° F. The efficiency of drying with superheated steam in an ordinary, non-jacketed mixer is only in the neighborhood of 50 per cent., and taking again 70 per cent. for the combined efficiency of boiler, superheater, and piping, we arrive at an ultimate efficiency of 35 per cent. for the whole steam-end of the installation. Consequently we have to use $56,040 : 0.35 =$ 160,000 B.t.u., equal to 12.7 lb. of coal with 12,600 B.t.u. per pound. This is 0.635 per cent. of the weight of briquettes. In most plants it is nearer to 1 per cent., because the coal-pitch mixture is generally heated above 80° C.

The grand total fuel-requirement for drying, heating, and for power (if electric current is not furnished from an outside source), will be illustrated by the following calculation:

Assume a coal with 15 per cent. of moisture to be dried to contain 5 per cent., 7.5 per cent. of pitch to be used, and the temperature of the mixture to be 210° F.; the original temperature of the cold materials to be 60° F.

We have $0.9Q = 1,850$, or $Q = 2,055$ lb. of coal with 15 per cent. water. The theoretical heat-requirement, exclusive of power, is then

$$205 \text{ water } (152 + 966) + 1,850 \text{ coal} \times 0.22 \times 150 + 150 \text{ pitch} \times 0.4 \times 150 = 299,240 \text{ B.t.u.}$$

With direct-heat drying or heating, an efficiency of from 65 to 70 per cent. can be safely expected under normal running-conditions, and consequently we have to expend from 428,000 to 460,000 B.t.u. If the furnace-fuel is again a coal containing 12,600 B.t.u. per lb., this is equal to from 34 to 37 lb. per ton of briquettes, or to from 1.7 to 1.85 per cent. of the weight of briquettes produced. The amount of coal necessary for generating steam for power-purposes is naturally very variable, depending upon the character and size of the machinery and the whole installation. From 4 to 10 h-p-hr. of engine or motor-

output are required for every ton of briquettes, or from 3 to 7.4 kw-hr. If not delving deeper into this subject, it will suffice to assume for a high-grade power-plant, operated under variable load-conditions, a total heat-consumption under boilers of 26,000 to 30,000 B.t.u. per kilowatt-hour, or from 78,000 to 222,000 B.t.u. per ton of briquettes. With coal of 12,600 B.t.u. as boiler-fuel, this is equal to from 6.2 to 17.6 lb. per ton of briquettes, or to from 0.31 to 0.88 per cent.

In view of the great variety of coals used in briquetting, it is preferable to express the fuel-requirements in heat-units, as a basis for comparison, and also to permit of quick calculation in a specific case. The total heat-requirement, including power, ranges from 110,000 to 245,000 cal. per metric ton, or about 400,000 to 882,000 B.t.u. per short ton of briquettes. With an average grade of coal this establishes a fuel-expenditure equal in weight to from 1.5 to 3.5 per cent. of the briquette-output. An inefficient power-plant, wet coal, and severe weather will demand more fuel, but most briquetting installations are within the 3.5, and 4 per cent. is generally considered the maximum. These data refer to coal-briquetting with hard coal-tar pitch as a binder.

Introduction of the Thomas Basic Steel Process in the United States.

Discussion of the paper of George W. Maynard, p. 280.

HENRY D. HIBBARD, Plainfield, N. J. (communication to the Secretary*):—This very interesting history is a valuable addition to our knowledge of the basic process and its introduction to this country. In addition to the attempts noted in the paper which were made here to establish the basic Bessemer process, and the experiment with the Siemens-Martin process at the Otis works, there were others made with the latter-named process at about the same date, all of which failed or, at least, were discontinued.

The first attempt which survived was that of Carnegie,

* Received Mar. 19, 1910.

Phipps & Co., who took up the manufacture of basic open-hearth steel at their Homestead plant. This departure was directly due to, and closely followed, a paper on the subject by J. W. Wailes, read before the Iron and Steel Institute in 1887.

The trial at the Otis works was made under the direct supervision of the late George W. Goetz, who told me of some of their experiences. The bottom was rammed in, as Mr. Wellman has said, of magnesite and tar, the rammers being heated red-hot to prevent the tar sticking to them. Mr. Goetz and a workman did the ramming. One effect of the hot rammers on the tar was to distill out some of the volatile hydrocarbons, and to make a smoke which so vitiated the air in the furnace that the men were nauseated and compelled to thrust their heads out of the furnace-doors, where they lay, vomiting copiously.

Shortly after that time I made my first basic open-hearth steel at the plant of the Linden Steel Co., of Pittsburg, now defunct. The operation was so troubled by silica from the sand on the pig-iron then available that, after a time, the furnace was relined with an acid bottom; though some of the results with the basic lining had been very good.

The trouble from the silica introduced into the slag, by the sand sticking to the pig charged, led me, the next time I tried the process, to melt the pig in another (acid-lined) furnace, and run it out on clean iron plates, merely to separate the sand. It also inspired later a paper read before the Iron and Steel Institute on the subject of sand on pig-iron, which apparently led to the production of machine-cast sand-free pig-iron.

Combustion in Cement-Burning.

Discussion of the paper of Byron E. Eldred, p. 479.

ROBERT SCHORR, San Francisco, Cal. (communication to the Secretary*):—In operations requiring the expenditure of fuel, it is solely a question of supplying a certain number of heat-units at certain temperature-conditions. The difficulty of controlling these temperature-conditions is mainly responsible for low efficiency, unless utilizable waste heat is carefully applied either for the pre-heating of the air necessary for combustion, or for the heating of the raw materials, or in other departments of the works to generate steam, etc.

A high theoretical or thermal efficiency very often does not establish a high "commercial" efficiency, and this fact is responsible for apparent crude practice in many arts. Coal is still very cheap in the East, and oil sells in the West for less than anywhere else in the world. Despite these favorable conditions, I believe that the list of references given in E. C. Eckel's book,[1] and also the contributions of C. Naske,[2] will show that Mr. Eldred's statement, that "the study of the phenomena of applied combustion has been neglected," is not justified as far as the burning of cement is concerned.

I have investigated the heat-balance of a lime-kiln and the commercial advantage of using generator-gas for calcination.[3]

Mr. Eldred proposes to calcine the materials in the present type of rotary kiln, gas-fired with long-flame combustion, and to take the fritted calcines to a special, gas-fired clinkering-kiln, which shall embody heat-regenerators. While his state-

* Received July 6, 1910.

[1] *Cement, Lime, and Plaster*, p. 511 (1907).

[2] *Zeitschrift des Vereines deutscher Ingenieure*, vol. xlix., No. 33, pp. 1354 to 1357 (Aug. 19, 1905); vol. l., No. 14, pp. 531 to 534 (April 7, 1906).

[3] *Engineering and Mining Journal*, vol. lxxxv., No. 12, pp. 613 to 615 (Mar. 21, 1908).

ments regarding temperature-requirements for calcining and for clinkering are correct, the difference between the heat-requirement for the present practice and for his mode of operation is less than 0.72 per cent., and the practical execution would be greatly in favor of the single rotary kiln, equipped with heat-saving arrangements. Should dust be troublesome in connection with regenerators or recuperators, it could be precipitated electrostatically with little expense. The superior economy effected by regenerators and by the utilization of the waste heat in other directions has been demonstrated in a number of American and European cement-plants; and there is no doubt that the introduction of further improvements would be greatly speeded by a more rapid increase in the price of fuel.

To illustrate the thermal conditions, I take Mr. Eldred's "theoretical cement-mixture," but I neglect almost entirely the heat generated by the combination of CaO and MgO with Al_2O_3 and SiO_2, because this item forms the only uncertain assumption. The results of experiments, which have been made in that direction by Le Chatelier and Berthelot,[4] differ widely, and Meade[5] even maintains that there is no heat liberated at all. The total range is an evolution of from 0 to 19,000 cal. for 100 kg. of clinker.

To permit of comparison, I will establish, first, the heat-requirement for an "ideal" kiln, that is, for the kiln with no loss by radiation and by incomplete combustion, and with a complete utilization of all waste-heat of the products of combustion, of the products of the calcination, and of the hot clinker.

The heat-requirement for dissociating CO_2 from $CaCO_3$ I have calculated[6] at = 43.3 for the molecular weight of about 100 = 0.433 cal. per g. = 433 cal. per kg. In the same way I get for the dissociation of $MgCO_3$, 17.9 for the molecular weight of 84, or 0.213 cal. per g. = 213 cal. per kg. I assume a lime-rock with 90 $CaCO_3$ + 4 $MgCO_3$ + 5 Al_2O_3, etc., + 1 H_2O = 100 per cent., and the other ingredients, clay and sand,

[4] *Le Ciment*, p. 12 (1905).
[5] *Chemical Engineer* (1906).
[6] *Engineering and Mining Journal*, vol. lxxxv., No. 12, p. 613 (Mar. 21, 1908).

to contain 5 per cent. of moisture. The composition of the raw-material mixture is then made up as follows: 122.32 of $CaCO_3$, 5.436 of $MgCO_3$, 1.359 of H_2O, 6.685 of Al_2O_3 and SiO_2 in lime-rock, 35.45 of clay and sand, dry, and 1.65 of H_2O in clay and sand; making a total of 172.90 kg. of raw materials to make 100 kg. of cement, or 648 lb. per barrel.

The quantity of flue-dust made should not exceed 2 per cent. Its specific heat, at 1,300° C., is about 0.21, which makes a heat-loss of $172.9 \times 0.02 \times 0.21 \times 1{,}280$ (outside temperature, 20°) = 930 cal., but only about half as much in good practice, because the temperature of the stack-gases will be about 600° C. In that event this loss amounts to 750 cal. per barrel, a very small loss, which will be neglected in my calculation.

The specific heat of the dry raw materials is 0.22. The average specific heat of hot clinker is 0.25; of CO_2 gas, 0.24; of superheated steam between 100° and $t° = 0.43 + 0.00037 \cdot \frac{100 + t°}{2} = 0.60$[7]; and of the products of combustion at high temperature is 0.265.

The last-named item will not appear in the investigation of the theoretically "ideal" kiln, because there the products of combustion discharge at the temperature of the atmosphere. It is further assumed that the calcination is done at 900° C. and the clinkering at 1,300° C.

The following calculations are submitted, and the results of the work of the kiln are given in Table I.:

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[7] Dr. Weyrauch, *Zeitschrift des Vereines deutscher Ingenieure*, vol. xlviii., No. 1, pp. 24 to 28 (Jan. 2, 1904).

TABLE I.—*Fuel-Consumption of Cement-Kilns Using Dry Mixtures.*

Class of Kiln and Mode of Firing.	Remarks.	Heat-Requirement (In Fuel). Per 100 Kg. Cement	Per Barrel, 170 Kg. Cement.	Per 100 Lb. Cement.	Per Barrel, 375 Lb. Cement.	Thermal Efficiency of Installation.	Coal of 7,500 Cal. Per Kg. (13,500 B.t.u. Per Lb.) Per 100 Kg. Cement.	Per Barrel Cement.	Per 100 Lb. Cement.	Per Barrel Cement.	Oil of 18,500 B.t.u. Per Lb. (Gallons, Lb. and Kg.) Per 100 Lbs. Cement.	Per Barrel Cement
"Ideal" Kiln..	....	Calories. 45,000	Calories 76,500	B.t.u 81,000	B.t.u. 303,750	Per Cent. 100	Kilograms. 6	Kilograms. 10.2	Pounds. 6	Pounds. 22.5		
Vertical Gas-Kiln....	In good German practice; sometimes better.	82,350	140,000	148,230	556,000	54.6	10.9	18.5	10.9	40.7		
Rotary Gas-Kiln ...	In excellent practice.	120,300	204,500	216,540	811,000	37.4	16	27.2	16	60.0		
Rotary Coal-Dust-Fired Kiln	In excellent practice.	148,200	252,000	266,666	1,000,000	30.4	19.7	33.5	19.7	73.7		
	In good, average practice, using long kilns.	185,250	315,000	333,333	1,250,000	25	24.7	42	24.7	92.4		
	In good, average practice, with 60- to 80-ft. kilns.	222,000	378,000	400,000	1,500,000	20.2	29.6	50.3	29.6	116.6		
Rotary Oil-Fired Kiln...	In best practice.	200,000	340,000	360,000	1,350,000	22.5	....	....	....	.. ..	19.2 lb. = 2.4 gal.	72 lb. = 9 gal.
											10°	Be'
	In average California practice.	244,444	415,555	440,000	1,650,000	18.4	. ...	. ...	.. ..	.. .	23.4 lb. = 3 gal.	88 lb. = 11 gal.
Rotary Gas-Kiln System with Regenerative System, as Proposed by B. E Eldred		90,000	153,000	162,000	607,500	50	12	20.4	12	45		
Probable Attainable Results with Rotary-Kiln Regenerator.................................. ...		72,800	123,700	131,040	491,400	61.8	97	16.5	9.7	36.3		

I. *Calcining and Clinkering in One Kiln at* 1,300° *C.*

1. Heat required for dissociating CO_2 from $CaCO_3$. 122.32 kg. × 433 cal. =	52,965 cal.
2. Heat required for dissociating CO_2 from $MgCO_3$: 5.44 kg. × 213 cal. =	1,158 cal.
3. Evaporating moisture and superheating resulting steam to 1,300° C.:	
(a.) for moisture in lime-rock: 1.359 [(100°–20°) + 536 cal.] + 1.359 × 0.60 × (1,300°–100°)	1,815 cal.
(b.) for moisture in other ingredients: 1.65 [(100°–20°) + 536 cal.] + 1.65 × 0.60 × (1,300°–100°) =	2,204 cal.
4. Heating the dry raw material to calcining-temperature of 900° C.: (172.9 – 3 H_2O) × 0.22 × (900°–20°) =	32,892 cal.
5. Heat required for clinkering: 100 kg. × 0.25 × (1,300°–900°) =	10,000 cal.
Total heat required for 100 kg. of clinker:	101,034 cal.

II. *Calcining at* 900° *C. and Clinkering in Separate Kiln at* 1,300°.

1. As above,	52,965 cal.
2. As above,	1,158 cal.
3. (a.) 1.359 [(100°–20°) + 536 + 0.60 (900°–100°)] =	1,489 cal.
(b.) 1.65 [(100°–20°) + 536 + 0.60 (900°–100°)] =	1,808 cal.
4. As above,	32,892 cal.
Total heat required for calcination:	90,312 cal.
5. Clinkering in separate kiln, which receives calcines at 900° C., as above,	10,000 cal.
Total heat required for 100 kg. of clinker:	100,312 cal.

Difference between I. and II. is 722 cal., or 0.72 per cent.

III. *Theoretically Utilizable Waste-Heat.*

1. In escaping CO_2: (122.32 × 0.44 + 5.436 × 0.523) × 0.24 × (1,300°–20°) =	17,407 cal.
2. In hot clinker: 100 × 0.25 × (1,300°–20°),	32,000 cal.
3. In flue-dust (neglected),	....
Total utilizable waste-heat per 100 kg. of clinker, say,	50,000 cal.
Total heat-expenditure per 100 kg. of clinker, about,	100,000 cal.
Actual, theoretic heat-requirement per 100 kg. of clinker, if deliberated heat is neglected:	50,000 cal.

Allowing only 5,000 cal. for heat deliberated by the formation of calcium and magnesium alumina-silicates, there will be required, for the "ideal" cement-kiln, 45,000 cal. per 100 kg. of clinker.

The theoretical heat-consumption of 45,000 cal. of the "ideal" cement-kiln installation with 100 per cent. efficiency, corresponds with a requirement of 6 lb. of coal, with 13,500 B.t.u.

per lb., for 100 lb. of clinker, which is equal to 22.5 lb. per barrel of 375 lb. (See Table I.) In this table will also be found data pertaining to actual kiln-performances in good cement-practice.

In connection with waste-heat saving-devices the question arises: What is the maximum efficiency that could be expected with such arrangements?

For the investigation of this matter let us consider the performance of the gas-fired kiln (No. 3, in Table I.), as the best obtainable with present equipments. There we have a fuel-expenditure equal to 120,300 cal. per 100 kg. = 811,000 B.t.u. per barrel = 60 lb. of 13,500-B.t.u. coal. Furthermore, we know that about 70 per cent. of the waste-heat, exclusive of radiation-losses, could be recovered for useful purposes.

Other Assumptions: Stack-gases, discharged at 600° C., of average specific heat = 0.265. A fuel to be used which will require a theoretical air-supply = 10 kg. per 1 kg. of fuel.

IV. *Total Utilizable Waste-Heat of Assumed Kiln.*

	Cal. per 100 Kg of Cement.
1. In products of combustion: $16 \times 11 \times 0.265 \times 600$ = about,	28,000
2. In escaping CO_2: $(122.32 \times 0.44 + 5.436 \times 0.523) \times 0.24 \times (600-20)$ =	7,875
3. In hot clinker (as above),	32,000
Total per 100 kg. of clinker:	67,875

V. *The Heat-Balance of Such a Kiln-Installation.*

		Per Cent.
1. Theoretical heat required of "ideal" kiln,	45,000 cal. =	37.4
2. Heat in products of combustion,	28,000 cal. =	23.3
3. Heat in products of calcination,	7,875 cal. =	6.5
4. Heat in hot cement-clinker,	32,000 cal. =	26.6
5. Heat-loss by radiation, etc.,	7,425 cal. =	6.2
Total,	120,300 cal. =	100

Items 2 + 3 + 4 = 56.4 per cent. If 70 per cent. of this could be utilized, or about 47,500 cal. per 100 kg. of cement, we have an actual heat-requirement of 120,300 — 47,500 = 72,800 cal., corresponding with a thermal efficiency of 68.1 per cent., or with 491,400 B.t.u. per barrel. Estimated for a fuel with 13,500 B.t.u. per pound, the consumption amounts to 9.7 lb. per 100 lb. of clinker, or to 36.5 lb. per barrel of cement.

If the recoverable 47,500 cal. were employed to heat both air and raw material to a temperature of $t°$ C. we get:

$$170 \times 0.22\, t° + 3\, H_2O\, [536 + 0.60\, (t°-20°)] + 10 \times 16 \times 0.25 \times t° = 47{,}500$$
$$\text{or, } 78.2\, t = 45{,}856 \text{ cal., or } t = 570° \text{ C.}$$

If we assume an efficiency of 80 per cent. for the gas-generator (inclusive of steam furnished to the same), we have $120,300 \times 0.80 = 96,240$ cal. actually delivered at the No. 3 kiln for every 100 kg. of clinker made, which corresponds to an efficiency of about 46 per cent. for the kiln itself, exclusive of the gas-generator end.

If the fuel-consumption could be reduced to 72,800 cal., as has been figured, only 58,240 cal. would be delivered at the kiln for every 100 kg. of clinker = 77 per cent. kiln-efficiency.

Returning to Mr. Eldred's proposal, he has a calcining-kiln with about 56 per cent. efficiency and a regenerator-clinker kiln of about 67 per cent. efficiency, the first taking 73.4 per cent. of all the fuel.

Consequently, we have a heat-loss of $0.44 \times 0.734 = 32.3$ per cent. in the calcining-kiln and $0.33 \times 0.266 = 8.78$ per cent. in the clinker-kiln installation, making a total loss of about 41 per cent., or a combined thermal efficiency of 59 per cent. for the complete installation, if 84 per cent. of the total heat of the gas used in the clinker-kiln is utilized. Such excellent results cannot be expected under practical running-conditions, and for that reason a maximum of 50 per cent. may be considered attainable.

Byron E. Eldred, New York, N. Y. (communication to the Secretary*):—Respecting the criticism of Robert Shorr on my paper:

I endeavored, and feel that I succeeded, in pointing out the advantages derived by the division of the burning process into two distinct steps, calcining and clinkering, to obtain the advantages of securing heat for regeneration for the second or clinkering step, which heat is not obtainable to a sufficiently economical degree in single-kiln processes. The waste gases of the single-kiln process are diluted to such an extent by CO_2 from the calcining process, and are further so exposed to the cooling influences of freshly-entered raw materials, that the benefit of regeneration is almost entirely lost. Furthermore, in the two-step process, the required velocity of heat-carrying gases is much reduced, giving the fuel a longer period for performing useful work and allowing for the use of producer-gas in the place of powdered fuel, which latter requires

* Received Sept. 21, 1910.

such a large amount of excess air to effect its combustion in suspension.

My paper was not the result of theoretical calculation alone, but was based on more than 10 years' practical experience and study of long-flame combustion and its application to producing localized combustion directly in contact with the material under treatment. I have found that a saving of 40 or 50 per cent. of fuel is easily effected by the application of the long-flame combustion to calcining operations, as compared with these operations performed by pregenerated heat, as in the case of present cement-practice. The calcining step in cement-burning is the heat-absorbing operation, and this step in present practice is not under independent control, as it must be to effect material economy. My suggestion of two-stage process insures absolute stage control, freedom of excess dilution of CO_2, and absence of dust in the regenerators. The two latter conditions are absolutely necessary for complete success in any regenerative system.

A Method of Calculating Sinking-Funds, and a Table of Values for Ordinary Periods and Rates of Interest.

Discussion of the paper of J B. Dilworth, p. 533.

Frank Firmstone, Easton, Pa. (communication to the Secretary*):—Mr. Dilworth's formulas and tables are of course correct, and the extinction will occur as calculated, provided the money periodically set aside for sinking-fund be really invested and the interest compounded as assumed in the calculation. Actually, in almost all cases, such funds are used "in the business," in which case, in addition to the periodical charges for principal of the fund, additional charges must be made at the end of each period to cover the interest and compound interest on previous charges. The periodical charge for sinking-fund thus becomes a varying and continually increasing one until the date of extinction. On many accounts it seems more natural and convenient to make the extinction by a fixed periodical charge under the legal rule for partial payments,

* Received July 27, 1910.

namely, that each payment should apply first to the interest due at the date of payment, and any remainder, after this is satisfied, be deducted from the principal before the interest for the next period is calculated.

The formula below is convenient for such calculations:

S, Principal to be extinguished.
Q, Periodical payment or charge for sinking-fund.
B, Balance unpaid at date of n^{th} payment, but before that payment is made.
r, Rate of interest per period per cent.
n, Number of payments or periods.
Then,

$$B = S\,(1 + r)^n - Q\,\frac{(1 + r)^n - (1 + r)}{r} \qquad (1)$$

If the debt is to be extinguished by the n^{th} payment, B must be equal to Q, and putting Q for B in equation (1) and solving for Q, we have:

$$Q = \frac{S\,(1 + r)^n}{1 + \dfrac{(1+r)^n - (1+r)}{r}} \qquad (2)$$

It is worth noticing that in the second member of (1) the first term is the amount of S for n periods at compound interest and the second term is the amount, also at compound interest, of an annuity, Q, for $n - 1$ terms.

About 10 years ago I had occasion to make a number of such calculations, and as I could find no method, in books easily accessible to me, but by a step-by-step calculation, I asked Prof. Mansfield Merriman for the titles of works in which I could probably find what I needed. Instead, he kindly wrote out and sent me the above formula, which I have never seen in print. It is easily verified by writing out the expressions for three or four consecutive periods, when the law of the series becomes manifest.

Crushing-Machines for Cyanide-Plants.

Discussion of the paper of Mark R. Lamb, p. 672.

HERBERT A. MEGRAW, San Luis de la Paz, Guanajuato, Mexico (communication to the Secretary*):—I am in entire accord with the basis of Mr. Lamb's argument, that the stamp

* Received Aug. 11, 1910.

is neither the best nor the most economical machine for primary grinding of ores for the cyanide process, and I believe with him that in the majority of cases other types of machines will do better work at lesser cost.

While it is dangerous to generalize on the basis of isolated data, we are gradually accumulating facts of the work done on similar ores by other machines, thus affording the opportunity for a comparison of data which will enable us to make intelligent deductions and decisions. A point of particular interest to me is the reference to the old-style Chilean mill. Under modern conditions and with intelligent regulation, these machines will do surprising work, although some modifications would make them more suitable for continuous hard work. The principle of the machine is very good, and I expect to see its field of usefulness broadened in the near future.

One of Mr. Lamb's statements, however, will be very generally challenged and I shall have to add my protest. He says, "In fact, except for the occasional manganese-silver ore, it might be said that cyaniding is a purely mechanical process, requiring a chemist merely as a form of insurance." Every one is likely to believe that his own specialty is the most important one to be considered, and into this error Mr. Lamb has fallen. It would be futile to attempt to record here the reasoning to prove the necessity of chemical knowledge in the everyday work of cyaniding. The process, based on chemical principles, was invented and developed by chemists largely, and the end of its chemical development has not yet been reached. It requires chemical knowledge to know when to add and when to omit reagents which may totally change the course and results of treatment. Chemical knowledge only, enables one to reach an intelligent conclusion without wasting time on theories which that knowledge shows unreasonable. Decidedly the chemist is necessary to the success of even the simplest cyaniding process, just as the mechanic is needed, and as power is needed to move the machinery.

Recent Progress in Blast-Roasting.

Discussion of the paper of H. O. Hofman, p. 739.

JAMES W. NEILL, Pasadena, Cal. (communication to the Secretary *):—Professor Hofman's paper brings the art up to date. As I was one of the pioneers in this business, I beg to give the following incomplete data regarding my efforts in this line.

In the year 1883, while I was in charge of the smelting-operations at Mine La Motte, Mo., there was at the concentrator an unusually large production of a middle product, locally called "iron," which carried considerable values in nickel and cobalt, and which it was desirable, for this reason, to treat by itself. The works could not spare a furnace for the purpose; and to handle this product I improvised a "roast-hearth" from an old "lead-hearth." This was a small cast-iron furnace of the usual construction, about 2 by 4 ft. in dimensions, with a hollow cast-iron back, through which the wind circulated before entering the tuyères. I had a cast-iron grate made to fit this hearth, and placed it flush with the lip, or apron, and above the tuyères. On this grate, which had round holes about 0.25 in. in diameter, and a total surface of about 20 by 40 in., I roasted the nickeliferous pyrite product; starting the fire with kindling, then adding charcoal, and then gradually feeding the sulphides with a shovel, so as to cover the fire, choke down blow-holes, and get an even bed. Blast at a pressure of 3 or 4 oz. was supplied by a fan-blower.

When a cake of roasted material had formed, it was removed with a steel bar, or fork, to the lip of the furnace, and thence into a wheel-barrow. The fine material which had remained unmelted, but very hot, on top of the cake would promptly ignite when the cake was broken up and removed. The fire would then be covered with fresh material and

* Received Aug. 29, 1910.

smoothed down, so that the fire was continuous, although the feed and discharge were not. I did not keep, in this operation, any accurate records of the relative amounts of product and crude material, since the method was slow in its development, and we had many days of poor luck, until we learned the tricks of the trade, and the best way to handle the stuff.

Among these discoveries was the necessity of keeping the crude concentrates as wet as possible, thus holding the material together and preventing it from running into and through the fire and grates. The wet material banks the fire down, keeping it confined to the lower stratum where it burns more fiercely; and it prevents blow-holes, etc.

All my records, assays, and analyses of this first experiment and of all the later ones described below, were destroyed in the San Francisco conflagration; so that I am obliged to give the results from memory, without details.

In 1889, while I was in charge of the Harrison Reduction Works, at Leadville, Colo., the change in the product of the mines from oxidized to sulphide ore began to make itself felt. The plant had no roasting-furnaces; but prices obtainable for treating sulphides left a much better margin of profit than those for other ores; and we tried to use as much of them as we could without undue production of matte and foul slags. Of course, when the sulphides were raw, this quantity, and consequently the margin of profit on the charge, was much less than it would have been on roasted materials; consequently, I rigged up one of the blast-furnaces as a "blast-roaster."

Bars were passed through opposite tuyères; cast-iron grates were placed on these bars, permitting the blast to come under the grates; the front was bricked up under the grate, to form a wind-box; and the front jackets were taken out to permit work on the grate. The hearth so obtained was about 33 by 60 in. in size. The materials were wheeled and dumped alongside the furnace, and the roasted and sintered material was hauled out and dumped into wheel-barrows on the other side. When the crucible under the grate had filled up with fines, the front was opened and the stuff was shoveled out. On this improvised hearth I roasted many hundred tons of ore, concentrates, etc., during a period of several months. All this material had been previously sampled; and the roasted material,

separated into coarse and fine, was also sampled, so that the results obtained were quantitative; and I am sorry that I cannot give them.

Many visitors came to see this furnace working. It was a pretty sight, when the charge was turned over, and the fine sulphides ignited and burned, like real fireworks. During this period I was notified by the attorney of F. L. Bartlett, of Canyon City, Colo., that I was infringing on the Bartlett patents; but Mr. Bartlett visited the plant, inspected the improvised furnace, heard my explanations and dates of previous operations; and I was not further molested in my operations. Mr. Bartlett's plant, using a somewhat similar furnace for the purpose of volatilizing zinc and lead sulphides and the making of pigments, etc., was in operation at Canyon City at this time; and his patents were probably the first to enter this field.

In the summer following my installation, John Williams, then manager of the Arkansas Valley smelter, at Leadville, installed a battery of Williams roasting-furnaces at the local plant, and issued a prospectus, setting forth their construction and method of operation and advantages. This furnace used a grate as mine did, but the roasted materials, when raked from the furnace, passed over a screen, and the coarse material went into a brick chamber under the upper grates, where it was subjected to a blast of air, with the object of carrying the roast to a further stage. As I remember the stated results, the coarse material carried from 7 to 9 per cent. of sulphur. This Williams plant consisted of several furnaces, and was operated regularly and for some time.

My next plant was erected in Bingham cañon in 1894 or 1895. I had secured a lease on a body of sulphide ore in the Commercial mine; and my sampling showed that this material had just about the grade then good enough for shipping; but as hauling,- freight,- and smelting-charges were high, I figured that by sinter-roasting, and thus reducing its weight by about 23 per cent., I could earn a handsome profit over the costs of roasting.

My furnace was constructed of cast-iron plates with two hearths, each 4 by 6 ft., divided by a heavy cast-iron plate. The blast entered through holes in the side-plates below the grates. The bottom was a hopper-shaped sheet-iron box, with a hole

in the bottom for the discharge of the fines; the smoke-hood was also of sheet-iron. There was no dust-chamber; the stack discharged straight into the air. This furnace was in all essentials the same as those erected later for the Yampa smelter and shown in Fig. 1. I roasted in it several hundred tons of ore from the mine, and a considerable amount of concentrates from the mill of the Old Jordan Mining Co., situated just above me in the cañon.

The ore was crushed through a 1-in. mesh screen; one man operated the furnace on each shift; and the tonnage was about 6 tons per hearth per day of 24 hr. The roasted materials, when drawn from the furnace, fell on a 1-in. mesh screen, and the fines were shipped separately. They carried about 15 per cent., and the coarse about 9 to 11 per cent., of sulphur; the amount of fines being 25 per cent. of the shipments, or about 17 per cent. of the crude ore. The pyrite-concentrates from the Old Jordan roasted very fast on account of the high sulphur; but the dust-loss also was high, as I had no dust-collectors.

During these operations I installed a small magnetic separator, over which I passed the semi-roasted fine material, and made a considerable shipment of the magnetic concentrates, which were of excellent grade. The installation was not a commercial one; but the results indicated a successful method.

As I remember it, the cost of roasting in this plant was about $1.15 per ton, not counting the steam for the blower, which was borrowed from an adjoining mill. Probably the total cost would have been $1.25 per ton. The operation of the plant was stopped by a sudden increase of smelting-rates in the Salt Lake valley. My product was not quite low enough in sulphur for the lead-furnaces, and the smelters did not like my teaching the miners how to profit by a roasting-operation; hence the "raise"!

This same furnace was sold, in 1902 or 1903, to the Boston & Montana Co., and erected at its smelter at Great Falls, Mont. I personally saw it started and "broke in" a couple of men to run it. The concentrates from Butte ores which were treated there did not carry as much sulphur as the Bingham materials; therefore the roasting proceeded much more slowly, and the percentage of fines produced was larger. Attempts to increase the heat by the addition of fuel, whether fine coal, coke, or

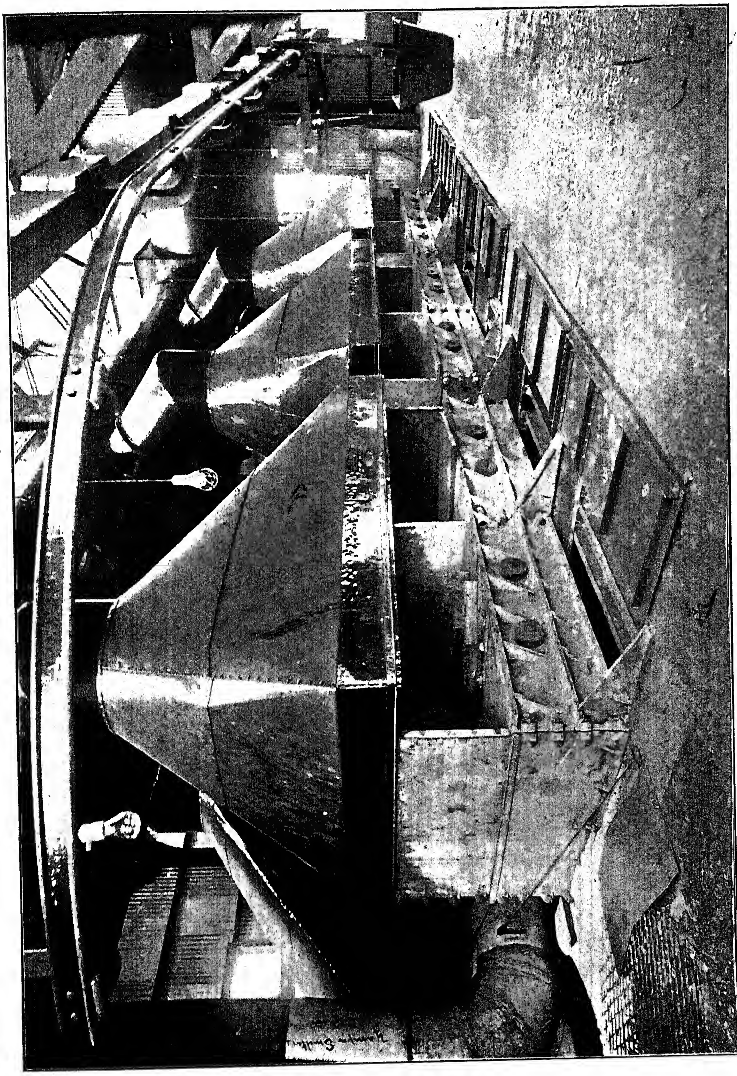
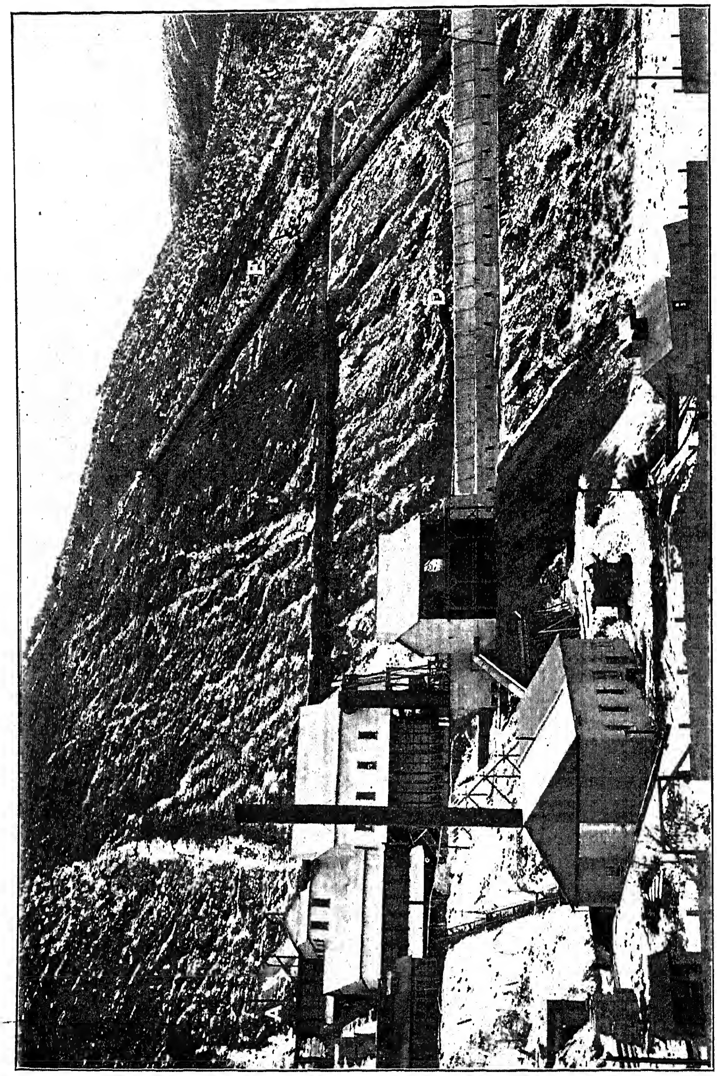


FIG. 1.—ROASTING-FURNACE INSTALLED IN 1903 AT YAMPA SMELTER, BINGHAM CAÑON, UTAH.



A. Crude-Ore Bin. B. Fine-Ore Bin. C. Roasting-House. D. Main Flue. E. Inclined Stack.

FIG. 2.—YAMPA SMELTER, BINGHAM CAÑON, UTAH.

charcoal, were not successful, since the ash, coke, or unburned coal in the clinker made it more brittle, thus destroying part of the value of the product. I do not know how much material was handled on the furnace at Great Falls. At the time of this installation, Mr. Lloyd (one of the patentees of the Dwight-Lloyd apparatus described by Professor Hofman) was General Foreman of the Great Falls plant; and he at once set to work to better the method of handling the material, so as to reduce the labor item, which, at Great Falls, was very high. One man could easily have attended two or three furnaces there; so that this item might have been cut in half. I think the costs at Great Falls were about $1.68 per ton of crude material.

Fig. 1 gives a good ideà of the furnace and plant erected by me at the Yampa smelter, in Bingham cañon, in 1903. There were six furnaces, each with two 5- by 6-ft. hearths, set upon a brick bin, which discharged into the charging-cars of the blast-furnace. The crude ore was brought on an overhead rail from the fine-ore bin, and dumped on the floor conveniently to each hearth; and the roasted material was raked through an opening in the floor into the bin below. The ore from the mine was to be crushed; then scťeened through 0.75-in. screens; the fines sinter-roasted; the coarse sent directly to the blast-furnace, to be concentrated by pyritic smelting into matte, which was to be shipped as flux.

Fig. 2 shows the general design of the plant. This roasting-plant was started under my supervision, one furnace at a time, so as to break in the crew, since the work requires at first considerable dexterity and judgment. By the time we had the crew half-trained, the roasted-ore bin was full; and as the blast-furnace was not ready to start, the roasting had to stop, and the crew were dispersed.

I was then called to Butte, and did not again see the plant in operation; but it was soon condemned and torn out, regular roasters and the roast-reverberatory method being introduced in its place.

During my first test here, three men operated the plant on each side, and one man brought the materials to the furnaces. The output was about 6 tons per furnace per shift, or 18 tons per day, and the cost of the first 275 tons (the first roasted bin-ful) was about 68 cents per ton of crude ore. But the work

was not well done, partly by reason of the inexperience of the crew, but more especially because the ore delivered to the plant was not up to the grade on which the furnaces had been planned, and which was promised. I had nothing to do with the mining.

It may be safely declared that such a roast-sintering hearth can be erected for about $2,000, and, of ore or concentrates carrying 35 per cent. or more of sulphur, will handle from 4 to 6 tons per shift, at a cost for labor and blast of about $1 per ton, when the crew have been properly broken in.

Of the product, 75 per cent. should ordinarily be fit for the blast-furnace and not carry over 10 per cent. of sulphur.

It was my purpose to try heated blast on one of these hearths for the purpose of accelerating combustion and the fusion of the material. Moreover, I believe the addition of heat through the blast would aid greatly in keeping the fire alive after the cake of sinter had been removed. In this operation there is such an excess of air blown through the charge anyhow, that the expansion through pre-heating, and the consequent dilution of the oxygen-content, should not make any trouble.

If the air were not heated above 300° or 400° F., it would not be too hard on the grates and other cast-iron parts of the furnace; and this amount of heat could be readily obtained in most plants, either from the stack-gases, or from reverberatory waste heat, or similar sources.

INDEX.

[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. References to papers expressly treating of the subject named are likewise in italics; and casual notices, giving but little information, are usually indicated by bracketed page-numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

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